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Vol. 106

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Sponsored by the
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Papers specially solicited for this volume by the Institute's Technical Committee on Reduction and Refining of Copper

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TO THE MEMORY OF

JAMES DOUGLAS

who devoted
most of his life to the
development of the copper industry
and to the well-being of engineers
this volume is dedicated

Preface

This volume is presented to the members of the American Institute of Mining and Metallurgical Engineers by the Committee on Reduction and Refining of Copper. A survey of the situation made in 1931 confirmed the belief that the metallurgy of copper had been losing some of its former importance in the Transactions of the Institute. It was apparent that sufficient material might be assembled to make a creditable volume, and the tentative plan was approved by the officers of the Institute. However, the quantity of good material in view was so great that it was considered impractical to devote the necessary proportion of the available publication funds of the Institute to a single subject. These facts were brought to the attention of the Institute's Rocky Mountain Income Committee. After reviewing the matter, the Committee, composed of Mr. Henry Krumb, *Chairman*, Mr. Harvey S. Mudd and Dr. George Otis Smith, recommended an appropriation to finance the publication of a special volume of Transactions under the sponsorship of the Rocky Mountain Fund.

There was a generous response to the Committee's request for special contributions, even though many authors were overburdened at the time by their own or their companies' activities. Particularly valuable help was given by Messrs. William Wraith, Frederick Laist and Heath Steele in authorizing and obtaining papers of vital importance in rounding out the volume. The effort has been made to cover each phase of copper metallurgy in an orderly and systematic way, so as to provide a book that gives a comprehensive description of the present status of the art.

Acknowledgment is made for their untiring work to the Vice-chairmen, Mr. C. W. Eichrodt and Mr. A. E. Wiggin; to Mr. G. L. Oldright, Secretary of the Committee in 1931, and Mr. A. B. Young, Secretary in 1932 and 1933; and to the other members of the Committee and subcommittees. The Committee appreciates the valuable advice and assistance rendered by the headquarters staff of the Institute.

Charles R. Kuzell, *Chairman of Committee on Reduction and Refining of Copper.*

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FREDERICK E. LAIST, *Chairman*
of Committee on Reduction and
Refining of Copper.

Career and Achievements of James Douglas

By Arthur L. Walker

James Douglas will long be remembered as a distinguished mining engineer and metallurgist, as a scientist with broad vision, as an executive whose understanding of practical problems coupled with his knowledge of men and affairs enabled him to weld together great mining interests and to their service build railroads and bring power. More, he will be recalled as a philosopher and a writer of charm, a historian of no mean order, and one whose catholic tastes and enthusiasms led him into the fields of education, medicine and theology.

His aim was to possess that which he considered made life worth while—friends and effectiveness in whatever task he undertook. In this aim he was successful to a degree exceeded by few, as thousands of his friends and associates could and can testify.

Dr. Douglas was born on November 4, 1837, at Quebec, Canada. His father, James Douglas, Sr., was a prominent physician and the leading surgeon of Canada. He also established the first retreat for the insane in the Dominion, the Quebec Lunatic Asylum, to which he devoted his services for years. The son followed in the footsteps of his father and studied medicine, not only in Canada but also in Scotland and in Germany. He entered the University of Edinburgh in 1855 but after two years at that institution returned to Canada and was graduated from Queens University at Kingston, Ontario, with the degree of Bachelor of Arts. Later he went to Laval University, Quebec, and subsequently became professor of chemistry at Morrin College, Quebec.

After he was graduated—at the time he considered his formal education completed—he traveled with his father in Europe and in the Orient; visiting Egypt, being especially interested in that country, several times. They gathered together an important archaeological collection which later was donated by them to the Metropolitan Museum of Art in the city of New York. James Douglas was also greatly interested in photography, a hobby in which he made use of his chemical knowledge. The writer can remember distinctly the descriptions he gave more than forty years ago in Arizona of the difficulties he encountered with the old wet photographic plates, especially while traveling in Egypt, when he prepared and used his own chemicals. In his humorous manner he told how he once discovered, in the nick of time, the nurse preparing to give the baby in the party a rub-down, using for that purpose the white, limpid

solution of nitrate of silver which had been placed in a bowl for photographic purposes. Only by his opportune interference was the transformation of the baby into an Ethiopian youngster prevented.

After traveling with his father, James Douglas returned to the University of Edinburgh, where he continued his course in medicine but devoted himself principally to the study of theology. He received his license to preach, a privilege that he exercised frequently, but was not ordained.

While young Douglas had specialized in medicine, theology and chemistry during his academic career, and had managed the lunatic asylum founded by his father, he had at the age of thirty-eight definitely decided to enter the ministry and devote his life to that work. Suddenly circumstances caused him to change his plans.

In 1875 he returned to Canada, where he discovered that his father had unfortunately invested heavily in gold-mining and copper-mining enterprises in that country, and was in danger of losing a large portion of his capital. The metal content of the material they contemplated working was too low to enable the mines to be profitably operated by any process then known to the world. Dr. Douglas abandoned his plans to enter the ministry and immediately turned his attention to the study of mining and metallurgy, with the fixed intention of entering that profession, in order to equip himself to rehabilitate his father's investments. The character of the mines precluded any chance of successful exploitation but the mining and metallurgical industry of the world has been enormously benefited by the influence this venture had on Dr. Douglas's career, for from that date his wonderful accomplishments in this branch of engineering had their beginning. His first venture in the metallurgical field was to associate himself with T. Sterry Hunt, a distinguished scientist, with whom he became acquainted while teaching chemistry at Morrin College. Together they worked out the Hunt and Douglas process, well known for many years, which was based on the solubility of copper oxide in ferrous chloride. After preliminary experiments at a property in Canada, Douglas was employed as superintendent, by the Chemical Copper Co. to take charge of its plant at Phoenixville, Pennsylvania, to apply the Hunt and Douglas process to the low-grade copper ore that the company was reducing. This plant, in which many difficulties were encountered, was not successful and later was destroyed by fire. It has been stated that the unsuccessful financial result could be attributed to high operating costs, for James Douglas, with his generous ideas, insisted on paying $1.50 per day for ordinary labor whereas 80 to 90 cents was the prevailing rate for a ten-hour day in Pennsylvania.

Then for a short period Dr. Douglas was out of employment. He visited Butte, Montana, several times and it has been publicly stated that he was the first to predict that the results of secondary enrichment would

be found in that area. While at Phoenixville he had received some carloads of copper ore from the Copper Queen mine at Bisbee, Arizona. This ore had attracted his attention and about 1880 he decided to make the long journey out there to look into a property that supplied ore sufficiently rich to stand shipment from the far West to a plant in Pennsylvania for treatment. The Copper Queen mine had been developed in a small way by Martin and Reilly but owing to the high rates for labor, the cost of and danger in bringing in supplies—the Apache Indians were often on the war path—and the difficulty encountered in smelting the ore in the one small furnace, blown in during the year of Douglas's visit, the operation of producing copper was not a financial success. James Douglas, with the experience he had gained and with his broad vision, which was of much greater importance, saw the enormous possibilities of this property and, having become acquainted with William E. Dodge, succeeded in interesting the latter and others in the firm of Phelps, Dodge & Co. in Arizona copper mines. He was retained by them to make a report on the Detroit Copper Company's property at Morenci, Arizona, and also to make a thorough examination of the original Copper Queen mine, a name that in after years became famous all over the world. Some of the partners of Phelps, Dodge & Co. had obtained an option on this property and they undertook to pay Douglas all the expenses for a thorough examination; and they agreed that if they purchased the mine they would put him in full charge, at the same time giving him an interest, contingent upon profits from the mine. It is needless to say that, on delivery of the report, the Copper Queen mine was taken over by Phelps, Dodge & Co.

A history of this property during the past fifty years would fill a large volume, and the story of the early days would be extremely romantic. The same is true of the three other copper properties in Arizona that came into being about 1880 or a year later, that of the Old Dominion Copper Co. at Globe, those of the Arizona Copper Co. and the Detroit Copper Co., both of which were situated in the Clifton-Morenci district. These mines were discovered by prospectors in search of gold or silver, as but little attention was given to copper. The original Old Dominion mine was located as a gold property because the narrow vein of copper ore discovered contained numerous flakes and particles of native gold, distinctly visible to the naked eye. All of these old properties, including the Copper Queen, had hard sledding during the first few years of their existence. The operating expenses were extremely high and the price of copper was low. Even with the rich ore they yielded it was difficult to make a profit. Then towards the end of the year 1887 the Société des Métaux, a French syndicate, under the management of M. Secretan, undertook to raise the price of copper and peg it at about 18 cents a pound. That was a godsend for the struggling Arizona copper producers

and during a period of about twenty months their operations were so profitable that they were able to improve their smelters, develop the mines and place themselves in a position to continue operating at a profit despite the low price to which copper went after the collapse of the French syndicate. In one of his numerous articles on the Copper Queen property James Douglas describes the elation he experienced when Secretan offered to contract for all the copper he could produce, and pay almost twice as much per pound as he had been receiving, considering shipping costs. While the Old Dominion, Detroit, and Arizona Copper companies continued to operate for many years, none of them ever rivaled the Copper Queen property. The upbuilding of this enterprise was due to the broad knowledge of the applied sciences, especially in the branch of metallurgy, which James Douglas possessed; to his keen perception, his ability as an administrator and as a handler of men; to the wonderful gift of broad vision; to his imagination, always necessary in a great enterprise, and to his financial acumen. He enjoyed the complete confidence of his principals and associates in the firm of Phelps, Dodge & Co., who became willing to back his judgment to the limit, financially and otherwise. While Dr. Douglas had many interests and received many honors, the development of the Copper Queen property into an enormous enterprise was his greatest achievement.

In operating the Copper Queen mine in the early days the first real difficulty occurred in 1884. The original orebody was about exhausted and finances were at a low ebb. Douglas stated orally shortly after that time, and since then has repeated it in his writings, that they were "drifting aimlessly in pursuit of others" (orebodies). The word "aimlessly" never seemed correct to the writer, for a new orebody was soon encountered by this drifting program; and success was due in a large measure to the geological knowledge Douglas had absorbed during his long academic career and the exact information he had obtained through his studies of underground conditions. Unfortunately for the Copper Queen, the Atlanta Mining Co., which owned the adjacent property, also was in distress and hunting for more ore. This company sank a shaft near the property line and by a strange coincidence this shaft uncovered the same ore the Copper Queen drift had entered; the discoveries were almost simultaneous. Then the inevitable legal questions hinging on the law of apex arose and the case was brought to the District Court. Dr. Douglas abhorred litigation, and, feeling that a decision had to be reached as to whether litigation or legitimate mining would be the principal industry of that district, exerted all the energy he possessed to effect a satisfactory settlement out of court. He was successful, as usual, and a consolidation with the Atlanta company was the result. Again James Douglas's keen vision was in evidence, for in developing the Atlanta property extensive orebodies were subsequently discovered.

He had a delightful sense of humor and in telling the writer about the events that happened during the course of the trial, he described how at one stage of the proceedings the experts for the Copper Queen picked up their reference books, reports and other paraphernalia and sedately marched out of court. The experts and lawyers on the other side were much mystified by this move and also greatly alarmed, as they were in a somewhat desperate position. This movement, Douglas said, with a twinkle in his eye, assisted in bringing the suit to a speedy conclusion and effecting the settlement.

In the early days the operating staffs of the copper-producing companies in Arizona were of the simplest character. James Douglas describes the staff of the Copper Queen mine, in an article written by him in 1913, in the following words: "For many years after the Copper Queen mine was opened the technical staff consisted of a Superintendent, an Assistant Superintendent (who was also the metallurgist), a mining engineer who did the mine surveying over the comparatively small area of worked underground with the assistance of a boy or rodman, and a chemist who was simply content to make determinations for copper, gold and silver."

As a matter of fact, however, the duties of the chemist were a little more arduous, because the composition of the slag had to be known by the metallurgist in order to prevent straying too far from a "type slag," as later it was called. Often the slag produced from smelting the oxidized ore mined at that time was far from having the proper composition, and assayed high in copper on that account. Coke was expensive and copper plentiful, so that relatively rich slags produced the best net financial results. As a result huge dumps of slag were accumulated, especially at the Copper Queen and Old Dominion smelting plants, which contained a vast amount of copper. But this copper was not lost, it was simply put in storage, for with the advent of the railroads and the discovery of sulfide copper ore in depth, to mix with the oxide ore, these slag dumps were resmelted with most satisfactory results. The old slag contained more copper per ton than had been reported, however. Dr. Douglas aptly summed up the situation when he said, "We never produced a slag containing more than 2 per cent copper but when we re-smelted it 3 per cent was recovered!" Evidently the chemists were philanthropically inclined and desired to make the metallurgists' path less thorny. At the plant of the Arizona Copper Co. at Clifton the slag was granulated and dumped into the San Francisco River, which flowed alongside the smelting plant. Though the cost of tramming slag from the furnaces to the dump by hand labor was saved, the copper contained in slag was lost forever. Dr. Douglas often used this as an example when strongly advocating the conservation of products and resources.

After obtaining control of the Atlanta Mining Co. Dr. Douglas acquired many of the other mines surrounding the Copper Queen, thereby assuring the future success of the property. This was about the year 1887. However, he was not satisfied with this group of mines, great as it had become, as a profitable mining and metallurgical enterprise. He investigated and finally obtained, by purchase or otherwise, mining properties in the Globe and Morenci districts in Arizona and in Nacozari, Mexico; and finally he extended operations into the Tyrone district in New Mexico.

The fuel problem, especially obtaining coke required for blast-furnace smelting, was of paramount importance to the Arizona copper producers in the early days. It was shipped long distances by railroad; then, in some cases, it was transferred to wagons of the old prairie type and hauled by mules or ox teams to the smelters. Prior to the building of a branch line from the Southern Pacific R.R. to Globe, the Old Dominion Copper Co. paid $40 to $50 a ton (of 2000 pounds) for coke. Even with this handicap the financial results were extremely profitable. Dr. Douglas was an expert in the field of transportation, always considering the future requirements of the enormous enterprise he was building up. He supplanted mule haulage from Fairbanks on the Southern Pacific R.R. to the mines and smeltery at Bisbee by traction engines, and afterwards built a railroad between these two points. Then the smelting operations were moved to Douglas, almost on the Mexican-Arizona line; and a great colliery and coking plant was established at Dawson, New Mexico, to supply the rapidly growing requirements for fuel.

Additional facilities were then required, so he planned and constructed the El Paso and Southwestern Railway system, which eventually operated over one thousand miles of track in connection with the Rock Island. By this means all of his plants and mines were provided with adequate transportation facilities and Dr. Douglas demonstrated his ability in the field of transportation as well as in mining and metallurgy. All of these properties and interests were later consolidated in Phelps, Dodge & Co., Inc., of which James Douglas was president until 1915, when he was succeeded by Walter Douglas, his son, though he remained Chairman of the Board till his death.

Dr. Douglas, like Andrew Carnegie, believed that scrapping old machinery was of paramount importance if industry was to progress. He applied modern methods and devices as soon as they had been proved and was the first to introduce in the Southwest electric power on a large scale in the mining districts and bessemer converters for copper matte. He innovated the use of wood for fuel in Loomis gas producer to supply the large gas engines installed at the Nacozari mine. He was the first in this country to employ the trough or barrel form of copper converter, and this type was frequently known as the Douglas converter.

Dr. Douglas abhorred secrecy in professional work. He was always willing to exhibit his methods and to share his ideas with others. He deplored the practice then in vogue, especially in nonferrous metallurgical plants, of the "closed shop." Those controlled by him were always open to inspection by any engineer, for it was his belief that "free trade" in ideas made for the advantage of all concerned. The campaign he conducted against metallurgical secrecy continued for years until it was crowned with success, due to his broadminded attitude and convincing arguments. One of the typical examples used by him referred to the decline of the mining and metallurgical industry in Swansea, where he said "they built a fence around themselves so high they could not see over the top of it." Had it not been for this plan of secrecy the operations in Swansea undoubtedly would have survived many years longer than they did.

In his presidential address to the members of the American Institute of Mining Engineers in 1899, on the subject of secrecy, Dr. Douglas stated that the "practice in the electrolytic (copper refining) plants was guarded with timorous care." Managers of the copper refineries during the nineties—of which the writer was one—will agree with that charge; and realize how many anxious days and sleepless nights might have been avoided had they been able to compare notes with each other. Eventually they all arrived at the same goal. In that address he also said: "The motives influencing the great body of writers who, without any pay, use the technical journals and such media of communication as our Transactions, in order to give to the brethren of their craft the results of their dearly earned experience are various and complicated, but in the majority of cases the impulse originates in the desire for reciprocity and in the hope that others will tell what they know in return for what we ourselves communicate and that, therefore, we shall learn at least as much as we can teach." The engineering profession and the world are indebted to James Douglas for the splendid and successful contest he waged against technical secrecy. The practice of reciprocity he introduced was largely responsible for the rapid increase in our industrial position in the world, as many writers and speakers have borne testimony.

Dr. Douglas was a writer of distinction even in his early years, and as time passed he became a voluminous contributor to the technical journals, principally on mining and metallurgical subjects but also on topics of general interest. He wrote many books, including several on historical and philosophical subjects. A complete record of his writings would cover many pages, but the following list prepared by the late Albert R. Ledoux, Past President of the A.I.M.E., illustrates the range of subjects covered by James Douglas:

Examples of Historical Volumes:

Canadian Independence, Annexation and Imperial Federation.
Old France in the New World.
New England and New France.
Journal and Reminiscences of James Douglas, M.D., by his son.

Selection from Articles, and Books:

The Copper Deposits of Harvey Hill, 1870.
Spectroscopic Observations of the Sun, 1870.
The Copper Mines of Chile, 1872.
Copper Mines of Lake Superior, 1874.
Metallurgy of Copper, 1883.
Cupola Smelting of Copper, 1885.
American Methods and Appliances in the Metallurgy of Copper, Lead, Gold and Silver, 1895.
Progress of Metallurgy and Metal Mining in America during the last Half Century, 1897.
Record of Boring in the Sulphur Spring Valley of Arizona, 1898.
Treatment of Copper Mattes in the Bessemer Converter, 1899.
The Characteristics and Conditions of Technical Progress of the 19th Century, 1899.
Gas for Use in the Manufacture of Steel, 1902.
Untechnical Addresses on Technical Subjects, 1908.
The Influence of Railroads of the United States and Canada on the Mineral Industry, 1909.
Earthquakes in Mines, 1911.
Development of the Railroads of North America and Their Control by the State, 1911.
The Copper Bearing Traps of the Coppermine River, 1913.

Dr. Douglas was an exponent of simplicity in writing and his book, "Untechnical Address on Technical Subjects," illustrates his idea in that respect. He wrote in a beautiful, lucid style. For example, in 1903, in an article describing the situation of the copper companies in Arizona, he stated, "The older companies are showing neither symptoms of decay nor of hysterical activity." What could be more explicit?

The members of the American Institute of Mining Engineers elected James Douglas President of the Institute in 1899 and again in 1900. In 1906 the gold medal was awarded to him by the Institution of Mining and Metallurgy of Great Britain. He received the degree of Doctor of Laws from McGill University in 1907. He was elected an honorary member of the Mining and Metallurgical Society of America in 1917 and in the same year was elected Chancellor of Queens College, of which institution and of McGill University he had been a trustee for many years.

Dr. Douglas was awarded the John Fritz Medal at the International Engineering Congress held at San Francisco in September, 1915, for notable achievements in mining, metallurgy, education and industrial welfare. He could not go to California to receive it, on account of his

health, consequently the presentation of the medal was made by Albert R. Ledoux on the fifth of December, 1915, at Dr. Douglas's home at Spuyten Duyvil, New York, at which ceremony he was surrounded by his children and grandchildren.

Dr. Douglas was keenly interested in the cancer problem and its alleviation and cure by the use of radium and X-ray. He visited the cancer centers of Europe in his study of this problem. He made a first gift of radium to Guy's Hospital, London. Later, in coöperation with Dr. Howard A. Kelly, of Johns Hopkins, he was instrumental in forming the National Radium Institute, and obtained from the carnotite ores of Utah over 12 grams of radium. This was divided between the Dr. Kelly's cancer center in Baltimore and the Memorial Hospital for the Treatment of Cancer in New York City, after sufficient for experiment had been placed with the Bureau of Standards in Washington. It was largely through his intense interest in cancer that the Memorial Hospital has become a leading cancer center of the world. He was liberal in his endowment and advice. There was founded then the Douglas Research Laboratory for research on cancer. His pleasure was giving: scores of deserving philanthropic efforts and deserving institutions were assisted and stimulated by him.

In 1905 Andrew Carnegie donated a large sum of money for a building to serve as the home and headquarters of the four Founder Engineering Societies, with the proviso that the ground on which the building was erected should be acquired by these societies. The A.I.M.E. found it difficult to raise the large amount of money required as its share until James Douglas came to the rescue. He volunteered to handle the financial obligation and to wipe out the debt incurred by the Institute. He was successful in this effort, largely because of his personal contributions and those of the members of Phelps, Dodge & Co.

Dr. Douglas died at his home on June 25, 1918, at the age of eighty-one. The funeral services were held in the Church of the Mediator, Kingsbridge, after which the body was taken to Quebec, the old home, for burial in the family plot. At a meeting following his death the Board of Directors of the American Institute of Mining and Metallurgical Engineers passed the following resolution:

> Through the death of James Douglas, this Institute, in common with the professions of mining and metallurgy, and the representatives of liberal learning, technical education, wise philanthropy and social progress throughout the world, are called to deplore the loss of an inspiring leader, tireless laborer, loyal and helpful friend.
>
> Dr. Douglas' sympathies, quick towards every worthy cause, were especially drawn towards the Institute, because its chief purpose, namely, the free interchange of professional knowledge and experience, commanded his lifelong allegiance, not only as a dictate of wise policy, but also as the result of an irresistible generous impulse. He gave freely; he gave "himself with his gift;" and his reward was

> known to all, even before the record of it, in the gratitude and grief of innumerable friends, was signed and sealed by his death. To them he was not only great, but dear.

Late in 1918 the Executive Committee of the A.I.M.E. recommended to the Board of Directors that the Institute establish a generous prize to be known as the Douglas Prize. A committee was appointed to determine the nature of this prize and the character of achievement for which it should be awarded. The committee recommended the establishment of the James Douglas Medal. The award was to be for "Distinguished Achievement in Nonferrous Metallurgy" in recognition of the attainments of James Douglas himself as a metallurgist. Eleven members of the Institute are proud to have been the recipients of this outstanding award.

In 1860 James Douglas married Miss Naomi Douglas, daughter of Captain Walter Douglas of Quebec. Six children were born, of whom the following survived him: Miss Elizabeth Douglas, Walter Douglas, Mrs. Edith M. Douglas, wife of Archibald Douglas, and James S. Douglas. All are still living. It is most remarkable that in each of these two generations a Douglas should have married a Douglas, and now in the third generation again a Douglas has married a Douglas. In all three marriages there has been absolutely no blood relationship. Such a record is unique.

In closing this brief biographical sketch, the writer, who went to Arizona in 1883 to accept a position with the Old Dominion Copper Co. at Globe and left there in 1893, deems it a privilege to record, in a few words, the debt he owes to this great man. While Dr. Douglas was investigating, and subsequently planning the operations for the Buffalo mines, he often came to Globe. During these visits many talks took place which were extremely pleasant and most profitable to the writer. The excellent advice given, in the most unassuming and casual manner, was of inestimable value, and the encouragement offered was inspiring in the extreme. The influence of his delightful personality and of his willingness to give unstintingly from his wonderful store of knowledge will never be forgotten. This is but one example typical of hundreds: it illustrates one of Dr. Douglas's many noble attributes—his desire to contribute to the welfare and advancement of his fellowmen.

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Smelting

History of Reverberatory Smelting in Montana, 1879 to 1933

By Frederick Laist,* New York, N. Y.

This paper is a review of Montana reverberatory smelting practice covering a period of approximately fifty years, during which time the small furnaces that had been in use elsewhere for a century or more, without much change in size or method of operation, were revolutionized and were developed into the huge, continuously operating furnaces of today. A study of Montana practice is particularly instructive, since reverberatory furnace smelting for the production of copper matte received more attention there than in other smelting centers and many of the improvements that finally resulted in establishing the preeminence of this type of furnace over the blast furnace originated in Butte and Anaconda.

The material in this paper was gathered from original sources, wherever possible, and all of the illustrations were made from original drawings, some of which have been simplified and worked over so as to be better adapted for reproduction. The aim has been to trace the modifications in design and operation which took place during the period in question—the reasons for the changes and the results achieved through them. For developments during the first twenty years, the technical literature of the period was freely drawn upon, but I am particularly indebted to W. M. Kelly, general foreman at the Anaconda Works, whose recollections and personal experience extend back to the early days. I wish to acknowledge the use made of publications of E. P. Mathewson, William Wraith and L. V. Bender, as well as information obtained here and there from friends and associates. The résumé of Great Falls practice was compiled from correspondence, notes and drawings found in the files of the old Boston & Montana Co. and the tabulations showing the performance of furnaces at Butte and Anaconda are adapted from plant records and correspondence in the files of the Anaconda Copper Mining Company.

The paper deals with some interesting comparisons that were made between furnaces operated under various conditions in connection with the Carson Case, and results obtained when using natural gas as fuel. This part of the paper may be considered original to the extent that these results have not previously been published.

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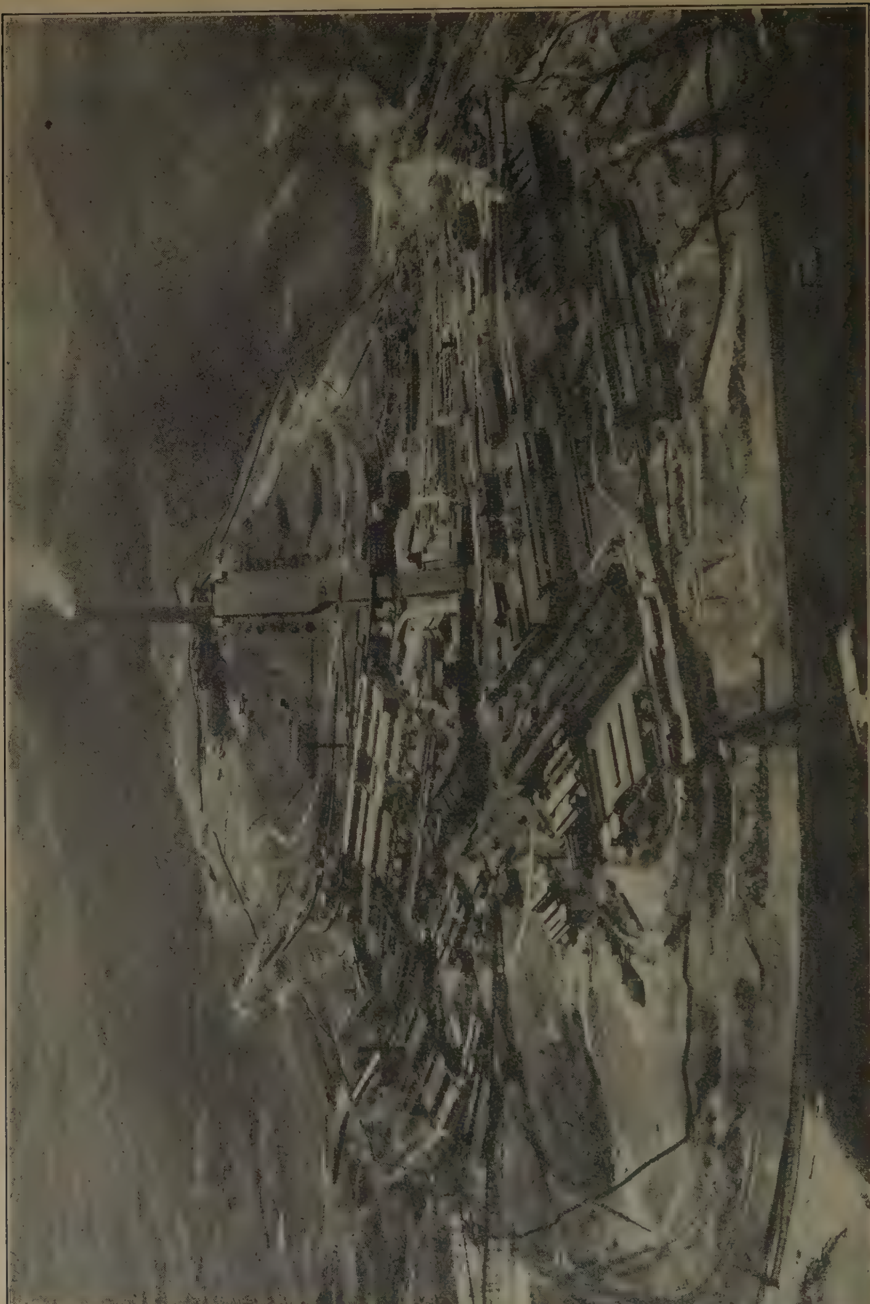


FIG. 1.—AIRPLANE VIEW OF ANACONDA REDUCTION WORKS, 1929.

Four Periods

It is helpful, in order to get a clear idea of the developments that took place during these years, to divide the time into four periods. The first started with the construction in 1879 of the first reverberatory smelting furnace to be built in Montana, and may be considered to have ended about 1890. During this period, numerous furnaces were built but their size and the general method of operation remained much the same as it had been in Wales, from which country the early Montana practice was borrowed. The second period extends from about 1890 to 1904. During this time, methods of operation were improved and furnaces were considerably enlarged but, in general, practice remained much as before. The third period commences with 1904 and ends about 1914. It witnessed the development of the long furnaces of today and further progress in the direction of continuous operation. Heating was, however, still done by coal burned in fireboxes. Oil had been adopted as fuel elsewhere but was not available in Montana. The fourth and last period starts with 1914 and extends to the present day. Firebox heating was replaced by coal dust and later by gas. Side charging was introduced and the practice of maintaining a deep bath throughout the length of the furnace was given up or modified.

Early Practice

The early smelting plants of Butte and Anaconda are briefly described by Professor Hofman and the equipment they contained is detailed. The first reverberatory furnace was built at Butte by Henry Williams at the plant of the Colorado Smelting & Mining Co. in 1879. This furnace had a hearth 14 ft. long by 9 ft. wide and used wood as fuel. It smelted about 10 tons of ore per 24 hr. and produced a matte assaying 60 per cent copper and from 700 to 800 oz. of silver to the ton. The matte was hauled 200 miles to the railroad and was shipped abroad. In 1880 two reverberatories with hearths 16 by 10 ft. and fireboxes 5 by 5 ft. by 4 ft. deep, were built at the Colusa smelter of the Montana Copper Co. Each of these smelted 12 tons of ore per day and burned 10 cords of wood. In 1881, six furnaces of about the same size were built at the Parrot Works. In 1883, similar furnaces were built by the Boston and Montana Co. The charges to these weighed from 2 to 3 tons and about 12 tons were smelted per 24 hr. to a matte assaying 60 to 67 per cent copper. An attempt was made at the Parrot Works to convert this to blister copper, but was given up on account of excessive cost. At first wood was used as fuel but later the fireboxes were adapted to burn Rock Springs coal, which increased the capacity of the furnaces to about six 3-ton charges, or 15 to 18 tons per day, with a ratio of charge to fuel of somewhat less than 2 to 1.

The operation of these furnaces may be considered typical of Butte practice of that time and is thus described by W. M. Kelly. Calcining was done in wood-fired, hand-rabbled reverberatory roasters. The

FIG. 2.—ANACONDA AND UPPER WORKS IN 1886.

calcine that accumulated in a pit at the front of each roaster was cooled and wetted by water to the extent of perhaps 10 per cent. A 3-ton charge of this wet calcine was wheeled to the side doors of the reverbera-

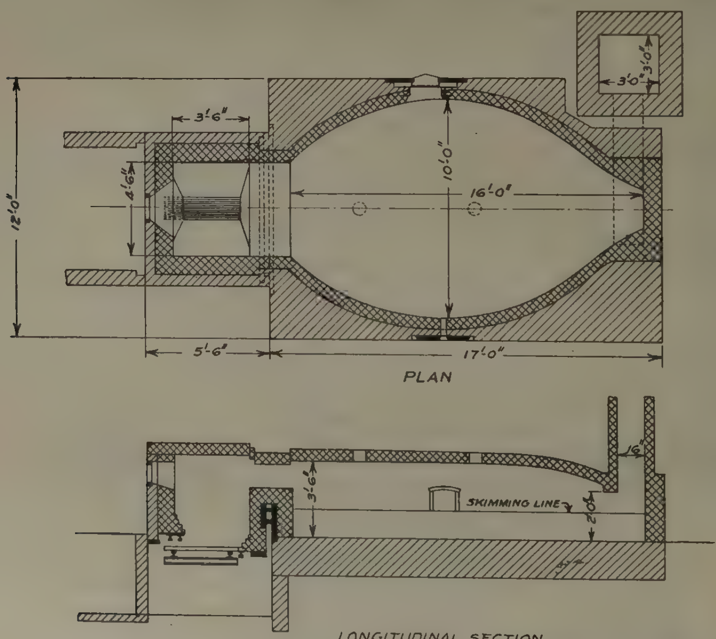


FIG. 3.—WOOD-BURNING MATTE FURNACE OF THE EARLY EIGHTIES.

tory and shoveled in by hand, throwing the charge against side and bridge walls and covering the surface of the bath. Coal was shoveled into the fireboxes by hand and slag was skimmed into sand beds where

it solidified into large slabs to be wheeled to the dump in barrows specially adapted to the purpose. After every third charge had been smelted, the matte was tapped into sand molds, cooled, sacked and shipped abroad. The inside of the furnace was then examined and damp, crushed quartz was shoveled, or placed by means of a long-handled paddle, against the corroded spots. The furnace was then ready for another cycle.

In 1883, construction was started on the Upper Works of the Anaconda Copper Mining Co., by Marcus Daly. Twenty-six reverberatory

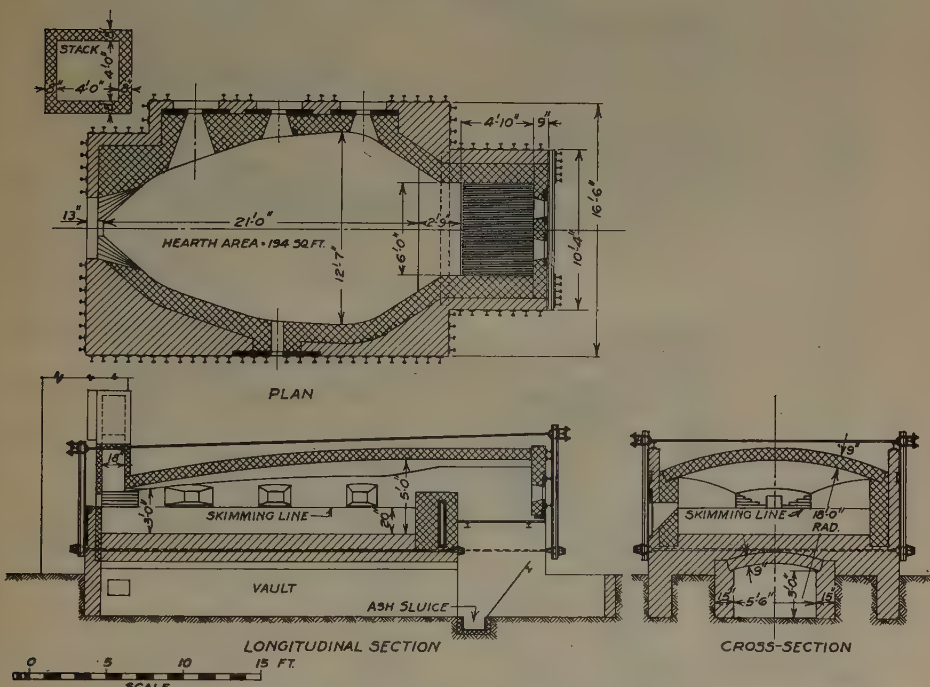
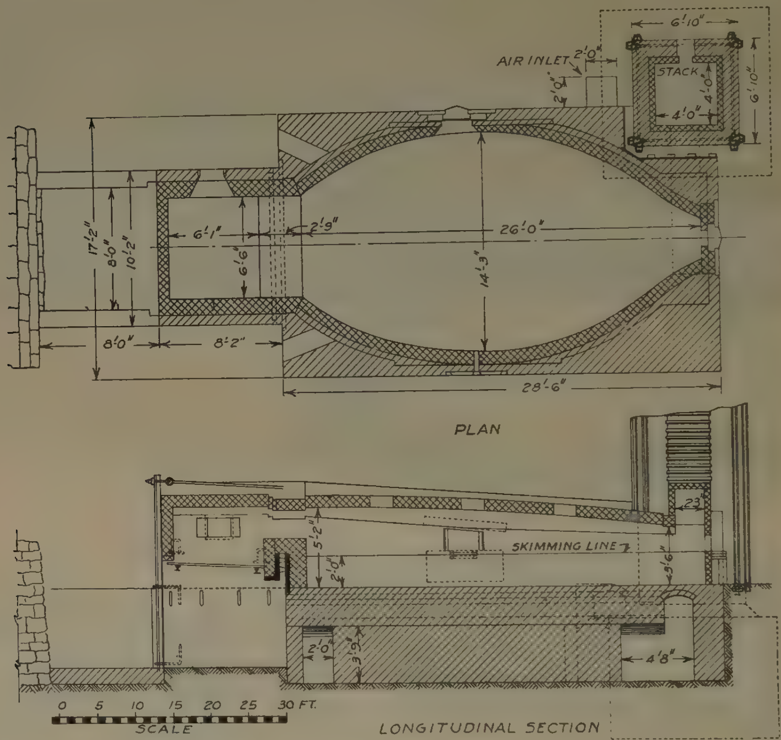


FIG. 4.—ANACONDA OLD WORKS MATTE-SMELTING FURNACE OF THE EARLY NINETIES.

matting furnaces, with hearths about 18 by 12 ft., were erected. They had individual stacks, smelted 3-ton charges and at first appear to have treated only about 15 tons per furnace per 24 hr. with a coal ratio of 1.5 to 1. The furnaces were arranged in two buildings. No. 1 contained 14 smelting furnaces and 12 hand-rabbled reverberatory roasters with one Bruckner cylinder. No. 2 contained 12 smelting furnaces and 12 roasters. Rock Springs coal was used for smelting, cord wood for roasting. Occasionally, when there was a shortage of coal, wood was substituted for part of the coal in the matte furnaces.

Two furnaces, 22 by 15 ft., were built in 1886 at the Butte & Boston plant. These appear to have been the largest in the district at that time. At Anaconda 29 new matting furnaces with hearths about 22 by 14 ft.

were built in the year 1887 at a new plant called the "Lower Works." Operations at this plant commenced in 1888 using Rock Springs coal for smelting. The furnaces treated 25 to 30 tons per day with a charge to fuel ratio when using Rock Springs coal of 2.5 to 1. The practice of



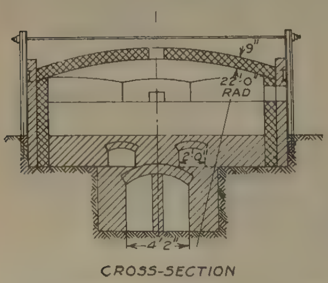


Fig. 5.—Reverberatory matte-smelting furnace, Butte, designed by H. C. Bellinger, 1893.

charging hot calcine was adopted here for the first time. The plant is described by W. M. Kelly as being divided into upper and lower floors. On the upper floor, there were 14 reverberatory furnaces and 36 Bruckner roasters. On the lower floor, there were 15 reverberatories and 60 Bruckners. All furnaces were connected through flues to three stacks on the hillside.

The Upper Works at Anaconda were rebuilt in 1890 when the furnaces were lengthened to 27 ft. and 40 Bruckner roasters were installed in place of the hand-rabbled furnaces. At the same time, arrangements were made whereby the calcines could be delivered hot, as they came from the roaster bins, into the reverberatories. The slag was granulated,

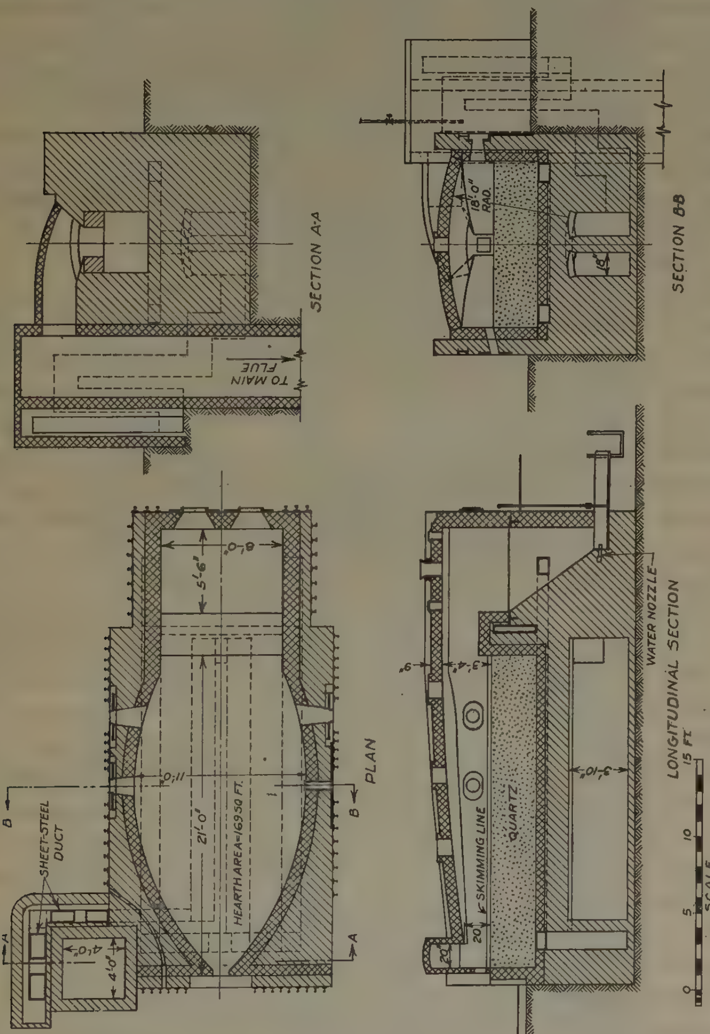


FIG. 6.—MATTE FURNACE WITH SPECIAL PROVISION FOR HEATING AIR AND BURNING HIGH-ASH COAL. CONSTRUCTED AT LOWER WORKS, ANACONDA, 1893.

elevated by means of a 30-ft. sand wheel and flumed to the dump. All furnaces were connected through flues to a 200-ft. stack on the hillside. The furnaces now smelted 30 tons per 24 hr. with a fuel ratio of about 2.5 to 1.

Mr. Kelly says: "The reverberatories were larger than the original furnaces at the Upper Works and operating practice differed very

materially. *Hot* instead of *wet* calcine was charged to these furnaces, and so far as I know, for the first time in the history of reverberatory smelting." The hot calcine from the Bruckners dropped into a brick pit or bin equipped with slide doors which permitted the hot calcine to be drawn into a 4-ton hopper car which could be raised, on an inclined tram by means of a hoist, to the floor over the furnaces. The calcine, after being dropped through a hopper on to the hearth, was spread by means of a rabble or spade to facilitate smelting. Before the installation of converters, the grade of matte was kept between 59 and 62 per cent. The Lower Works were destroyed by fire in March, 1889, were rebuilt, using structural steel instead of wood in the buildings, and started operations again in the fall of the same year. There were no material changes in design other than the substitution of steel for wood, and methods of operation remained the same.

The early furnaces were built over an open space, through which air was caused to circulate with the idea of keeping the bottom cool. This space was arched over and the hearth of the furnace was laid in the shape of an inverted arch enclosed by side walls of firebrick which supported the roof. A lining of firebrick, which could be renewed without interfering with the roof arch, protected the walls from damage by the flames and molten charge. The hearth was an elongated oval although the external shape of the furnace was generally rectangular and was strongly braced by steel I-beam buckstaves and tierods. The bridge wall between the hearth and the firebox was built around a "conkerplate," which served the double purpose of strengthening the bridge wall and keeping it from getting too hot, air being circulated through the channel in the "conkerplate." The arch was constructed of silica brick and generally had a rise of about one inch in the foot. It sloped rather steeply toward the front of the furnace until it approached within 12 or 14 in. of the skimming plate. The shape of the roof and of the hearth, as well as the size of the space where roof and side walls converged—technically known as the "verb"—were supposed to have much to do with the fuel efficiency and smelting rate of the furnace. At first it was considered important for each furnace to have its own stack, which was generally 4 to 5 ft. square and about 60 ft. high, but about 1887 the practice of connecting into a common flue leading to a single stack was adopted.

The hearth was constructed of ground quartz containing approximately 97 per cent silica. The smelting in of the sand hearths was done with utmost care in order to avoid—in so far as possible—the ever-present danger of having parts of the bottom floated up by the molten matte resting upon it. The quartz or sand, sometimes mixed with 3 or 4 per cent of slag, having been shoveled into the heated furnace, was rabbled, while the moisture and carbon dioxide were driven out and, as the temperature increased, was formed into the proper shape by means

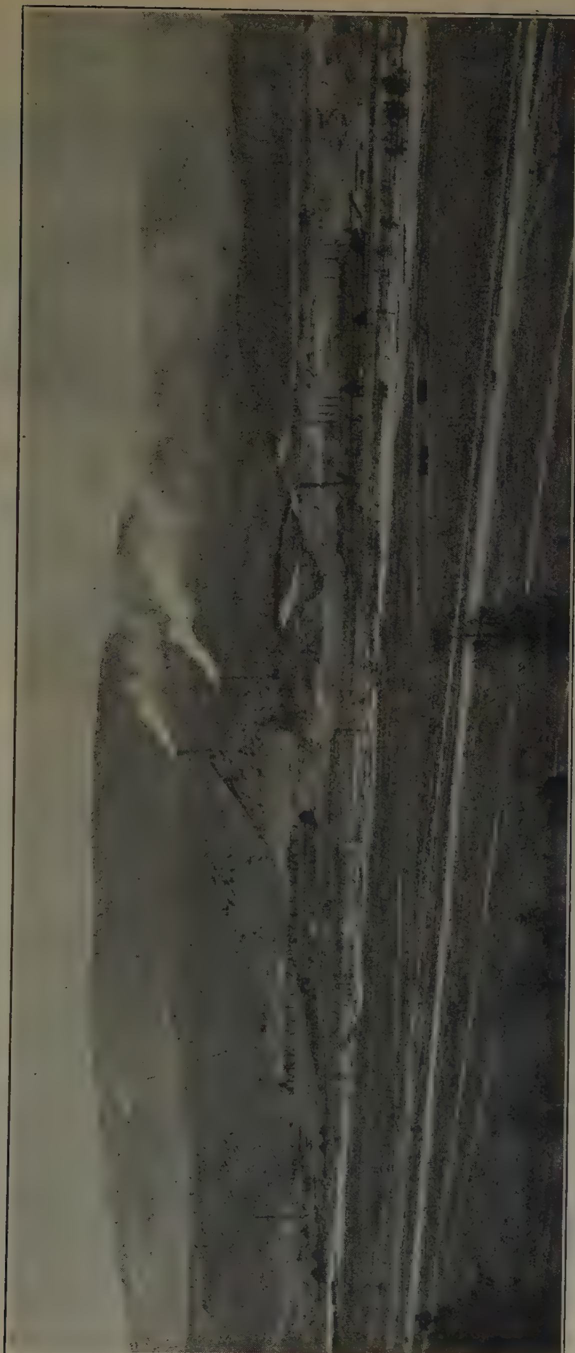


FIG. 7.—UPPER WORKS AND REFINERY, ANACONDA, 1899.

of long-handled paddles. The doors were closed and the temperature was gradually increased to the maximum degree obtainable and main-

FIG. 8.—MATTE FURNACE, LOWER WORKS, ANACONDA, IN THE EARLY NINETIES.

FIG. 9.—CHARGE FLOOR, UPPER WORKS, ANACONDA, IN THE EARLY NINETIES.

tained thus for several hours. The furnace was then allowed to cool, after which slag, crushed to about one inch size, was spread over the surface, fused and absorbed by the porous sand bottom. A second similar fusion

completed the saturation of the lower hearth, after which the sand was thrown in for the upper hearth. This was calcined and very carefully shaped up, fired, cooled, saturated and generally smelted in very much like the lower hearth. Finally the furnace was fettled and was then ready to go on its regular work.

The firebox was lined with firebrick and the grate bars were usually ordinary iron rods about one inch square. The relation between the areas of firebox and hearth was obviously of great importance. At first the tendency was to limit the area of the firebox while gradually enlarging the hearth, on the theory that a better utilization of heat would be thus obtained and less coal would be used per ton of charge smelted. Thus the small furnaces of 1880 with hearths about 10 by 15 ft. had fireboxes 4 by 5 ft., giving a ratio of approximately 1 to 5 for the areas, whereas the 35 by 14-ft. furnaces of a later date had 5 by 8-ft. fireboxes, giving an area ratio of about 1 to 9. In the long furnaces which followed, the ratio was about 1 to 18, but most of the smelting was done in about one-third the length of the hearth. It was soon realized that better results could be obtained by forcing the furnaces and by increasing the rate of combustion. The idea gained ground that fuel should be burned as rapidly and intensively as possible so that the flame temperature might be as high as possible in a comparatively limited area near the bridge wall where most of the smelting was done. This led away from the exaggerated oval shape of hearth, the theory of which appears to have been that room for expansion of the gases must be allowed immediately after they left the firebox. Instead, the furnaces were built with more elongated hearths and relatively wider fireboxes, which led to an intensification of the heat in the region near the firebox. This tendency reached its culmination in the long furnaces of 1905 in which the firebox was almost as wide as the hearth and the length of the hearth was six times its width, as compared with one and one-half times in the early furnaces.

Improvements in Operation

The character of the charge was well adapted to rapid driving as it was readily fusible and the matte fall was heavy, being 25 to 30 per cent of the total weight of charge. At Butte and Anaconda the calcined concentrates contained about 15 per cent copper and a slag running about 37 per cent silica and about 6 per cent alumina was ordinarily made with a matte ranging from 50 to as high as 65 per cent copper. The favorable character of the charge, together with the practice of charging hot calcines and other improvements in operating technique, led to the attainment of unprecedented capacities. The furnaces of the Montana Ore Purchasing Co., which were built about 1893, and those of the Butte & Boston Mining Co. of about the same date, are said to have treated

as much as 50 tons per 24 hr. and new records were constantly being established.

The old idea that the bottom of a reverberatory must be kept cool by circulation of air was being abandoned, and the vaults under the furnaces were being filled with slag. It is difficult to understand why that part of a reverberatory which is hardest to keep hot should have been deliberately cooled, but such had been the custom for generations and ideas firmly rooted in the past die hard. It is to the credit of the Montana metallurgists that they recognized the anomaly and had sufficient enterprise to break away from it. Gradually the modern practice of constructing the furnaces on a solid foundation became universal.

Probably the most important improvement in operating technique, apart from using hot calcine, was the maintenance of a large pool of matte in the furnace at all times. The idea had long been prevalent that matte was injurious to the bottom and that the furnace must be entirely emptied at frequent intervals in order to enable the workmen to inspect and repair the hearth as well as fettle the side walls. Then it began to be noticed that a pool of matte not only did no damage to the bottom but served as a protection and greatly aided the smelting of the succeeding charge. Steps were taken to prevent the workmen from tapping out the matte beyond a certain point and the practice was instituted of charging more frequently and skimming less completely so as to make the operation more continuous and less intermittent.

We begin to recognize the practice which immediately preceded that of today, and which was distinguished by:

1. A large pool of matte which covered the entire furnace bottom from one end to the other and had a depth of from 12 to 24 inches.

2. The charging of hot calcines at frequent intervals and in relatively small quantities.

3. The periodical partial removal of slag in such a manner as to cool and interrupt the steady running of the furnace as little as possible.

4. Fettling less frequently—sometimes only once a month.

5. Using the furnace as a storage reservoir for matte from which the converters obtained their requirements as needed.

The maintenance in the furnaces of a large quantity of matte was of particular importance when firing was done by means of coal burned in fireboxes. These fires required grating about every four hours, during which most of the evolution of heat stopped and the laboratory of the furnace cooled off considerably. The latent heat stored up in the bath served as a stabilizer, as the temperature of the bath did not go down with the temperature of the smelting chamber. Thus when the fresh coal, dropped into the firebox after grating, started burning again, it did not take long to reestablish normal conditions. The body of matte, or rather of matte and slag, held in the furnace at all times was very large, and

increased with the size of the furnaces, from 10 or 15 tons to as much as 150 tons in the long furnaces which came in about 1905. As will become apparent later on, the large pool of matte is no longer of such importance.

Another useful purpose of the bath was that it helped to spread the charges of hot calcines and prevented their adherence to the bottom. The molten, half-metallic bath was an excellent conductor of heat, and heat radiated from the roof and side walls and absorbed from the flames and hot gases got to the bottom through the bath more readily than through a thick layer of poorly conducting charge. The hot calcines, on account of the sulfur dioxide which they were constantly giving off, tended to flow like a liquid, and this tendency was increased when they were dropped on to a molten bath of relatively high specific gravity. The consequence was that it was no longer necessary, as in the early practice, to level off the charges, and the time and heat losses that accompanied this operation became negligible. The furnaces could be built with fewer doors and openings, thus decreasing air leakage, and they could be made of larger size, since the limitation of man power was no longer a controlling factor.

As the furnaces became larger, the bath and the capacity of the smelting chamber kept increasing in relation to the size of the charge. The early 10 by 15-ft. furnaces were operated on what might be described as a "batch" system and originally the size of a charge was the quantity of ore that would fill the smelting chamber but still leave room for the passage of the flames and hot gases. Fresh charges were introduced until the desired depth of matte and slag had accumulated on the hearth. The slag, being lighter, floated on top of the matte, and when the final charge of a cycle was completely smelted the slag was skimmed off and the matte was cast into sand molds. The furnace was then recharged and a new cycle was commenced. It is easy to see how such a system made it almost impossible to greatly enlarge furnace units and increase their efficiency. It is interesting to note in retrospect the reluctance with which seemingly obvious innovations, such as charging by means of hoppers, were received. The following excerpt from Peters (edition of 1892) illustrates the point:

> It is more advantageous to charge a furnace by the side doors than by means of a hopper above the roof, as the proper leveling and distribution of the heavy charges now used, are almost impracticable by means of the rabble, while when charged with a shovel, every pound of ore can be thrown just where it is needed. In order that no time be wasted, the helpers from other furnaces assist in charging—at least four men being required. The work is exceedingly hot and laborious, as the entire process should be completed in from 10 to 15 minutes, to avoid waste of time and fuel.

It thus becomes clear that along with the enlargement of the furnaces there had to go radical changes in practice, which would make the operation of the furnaces less dependent upon the strength and endurance of

Fig. 10.—Converters and Blast Furnaces, Lower Works, Anaconda, 1899.

Fig. 11.—Concentrator and smelter, Anaconda, Lower Works, 1899.

the individual workman, and to what extent this was accomplished the reverberatories of today bear abundant witness. The work of the men altered in character and it became possible for the same crew to attend to a furnace of greatly increased size and capacity while at the same time the physical labor required of them became very much less. The way having been thus prepared for enlargement of furnace units, the work

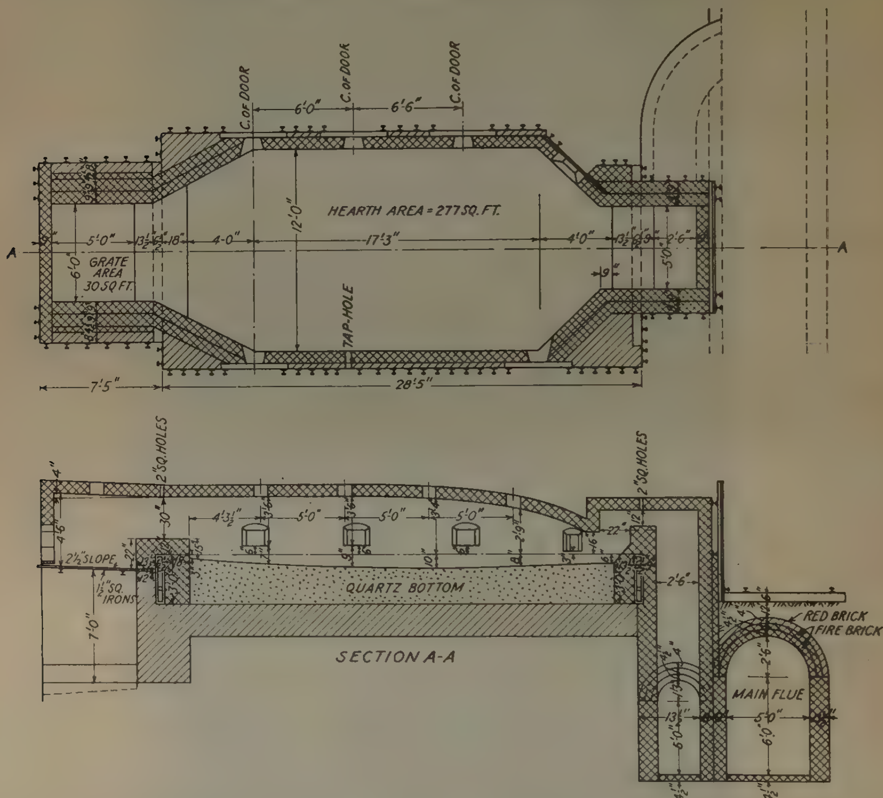


FIG. 12.—RECTANGULAR REVERBERATORY FURNACE DESIGNED AT ANACONDA IN THE LATE NINETIES.

proceeded rapidly and the decade from 1890 to 1900 witnessed a growth in hearth dimensions from 15 by 22 ft. to 20 by 50 ft., the Butte & Boston Mining Co. being again the pioneer with a furnace of this size, constructed according to drawings made at the Colorado Smelting Works.

From 1896 to 1900, inclusive, there were erected at the Butte Reduction Works three furnaces. The first of these had a hearth 16 by 35 ft. and smelted 45 to 50 tons per 24 hr. A second furnace, 16 by 45 ft., was built in 1897 and smelted from 60 to 70 tons per 24 hr. Finally, in 1900, a third furnace, 20 by 50 ft., smelted from 85 to 100 tons per 24 hr. At the plant of the Montana Ore Purchasing Co., by 1900 six furnaces were being operated with hearths 16 by 33 ft. At Anaconda four furnaces

were enlarged in 1899, to approximately 14 by 35 ft., and one to 14 by 45 ft. In 1901, 14 furnaces with hearths 20 by 50 ft. (later narrowed to 19 ft. by thickening the walls) were built at a new plant called the Washoe

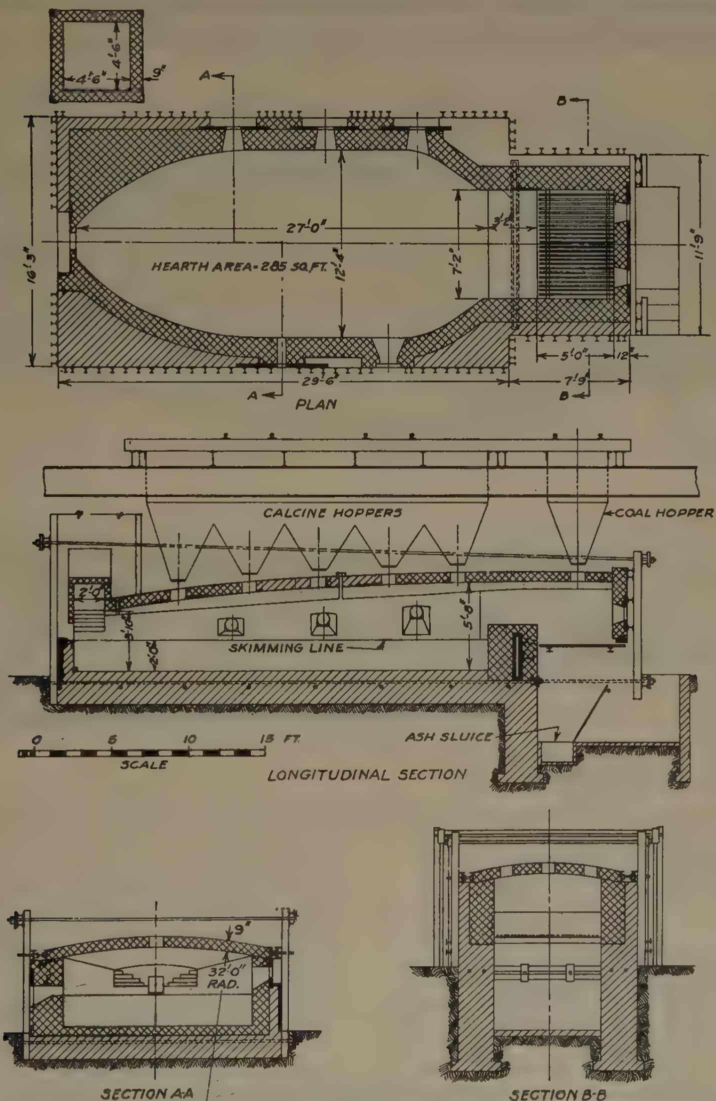


Fig. 13.—Typical Anaconda Old Works reverberatory matte-smelting furnace of the late nineties.

smelter, subsequently renamed the Anaconda Reduction Works. These formed the nucleus of the long modern furnaces.

While these enlargements were being made, a great deal of experimenting was going on looking toward increasing furnace capacity by a

more intensive combustion of the fuel. Ordinarily this was accomplished by enlargement of the firebox, the area and height of stack and by abandonment of the so-called "clinker grate" of the Welsh smelters. There were also, however, numerous attempts to use forced blast under the grates and at the Colusa smelter three 12 by 20-ft. furnaces had been

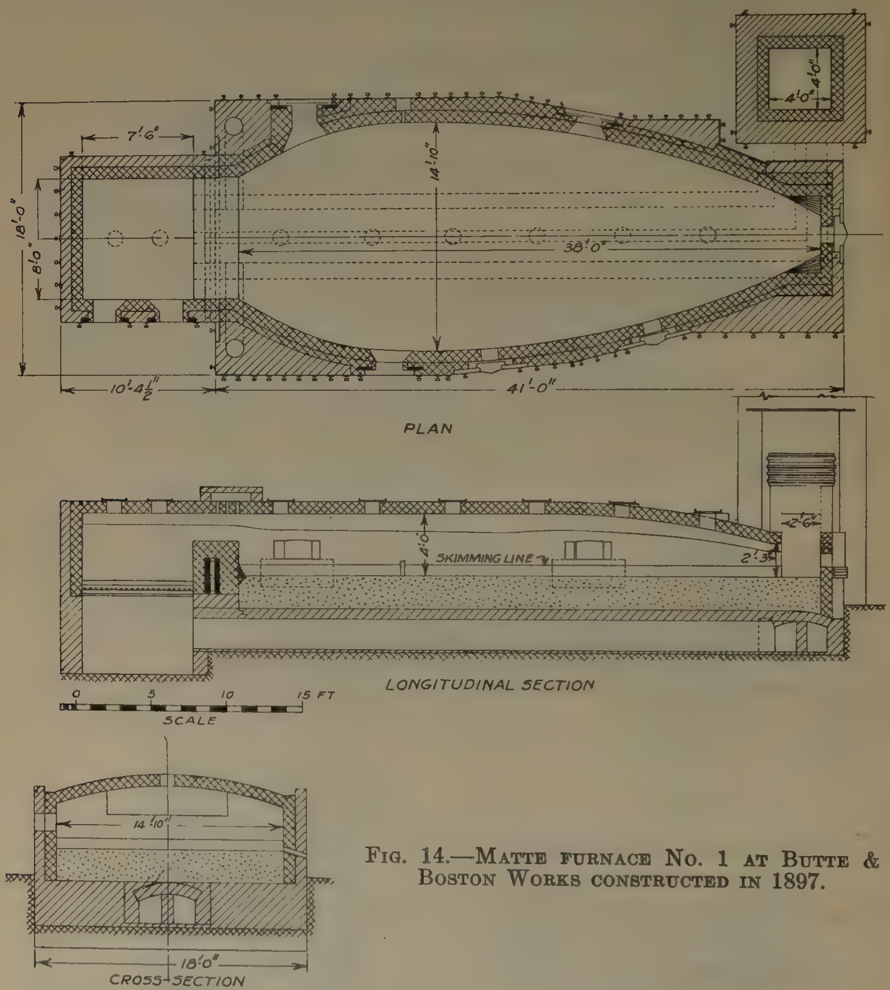


FIG. 14.—MATTE FURNACE NO. 1 AT BUTTE & BOSTON WORKS CONSTRUCTED IN 1897.

so equipped and were being operated with under-grate blast when the plant was closed down in 1899. These three furnaces averaged 135 tons per 24 hr., or about 45 tons per furnace day, which, considering their small size, was very good indeed. Similar experiments had been carried on at Anaconda but natural draft was preferred at the "Old Works." Nevertheless when the "New Works" were planned, forced under-grate blast was adopted. The system was not successful, perhaps on account

of the excessive clinkering of the Diamondville coal with which the furnaces were heated, and after every effort had been made to get the furnaces to work, the system was given up in favor of natural draft.

Attempts were also made to preheat the air used for combustion and Peters describes such a furnace built at Anaconda, which was designed to

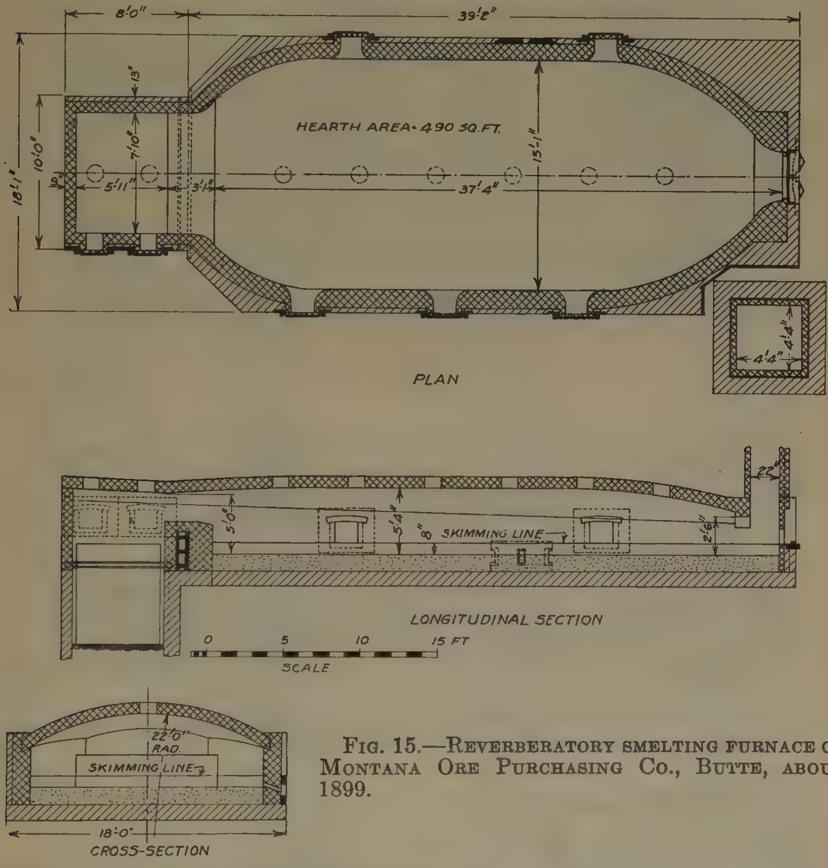


Fig. 15.—Reverberatory smelting furnace of Montana Ore Purchasing Co., Butte, about 1899.

use an inferior coal containing upwards of 20 per cent ash. The coal was fed from a hopper directly into the firebox and the ashes were sluiced away by a stream of water. The ashpit was closed by means of iron doors and the grates were shaken by means of a handle on the outside. The stack, starting about 6 ft. below the ground line and extending 15 ft. above, had double walls. The space between these walls was traversed by a zigzag pipe. The air required for combustion was drawn through this pipe, passing thence backward and forward through the main vault under the furnace and finally reached the firebox through large cast-iron conduits on either side of the ashpit. The air is said to have been heated to a temperature of 371° C. and the furnace, the hearth of which measured

only 11 by 21 ft., is said to have smelted 46 tons of hot calcines per 24 hours.

We have not been able to obtain confirmation of this statement as to tonnage smelted, although such a furnace was undoubtedly built. It is possible that a better grade of coal was used when smelting at the rate

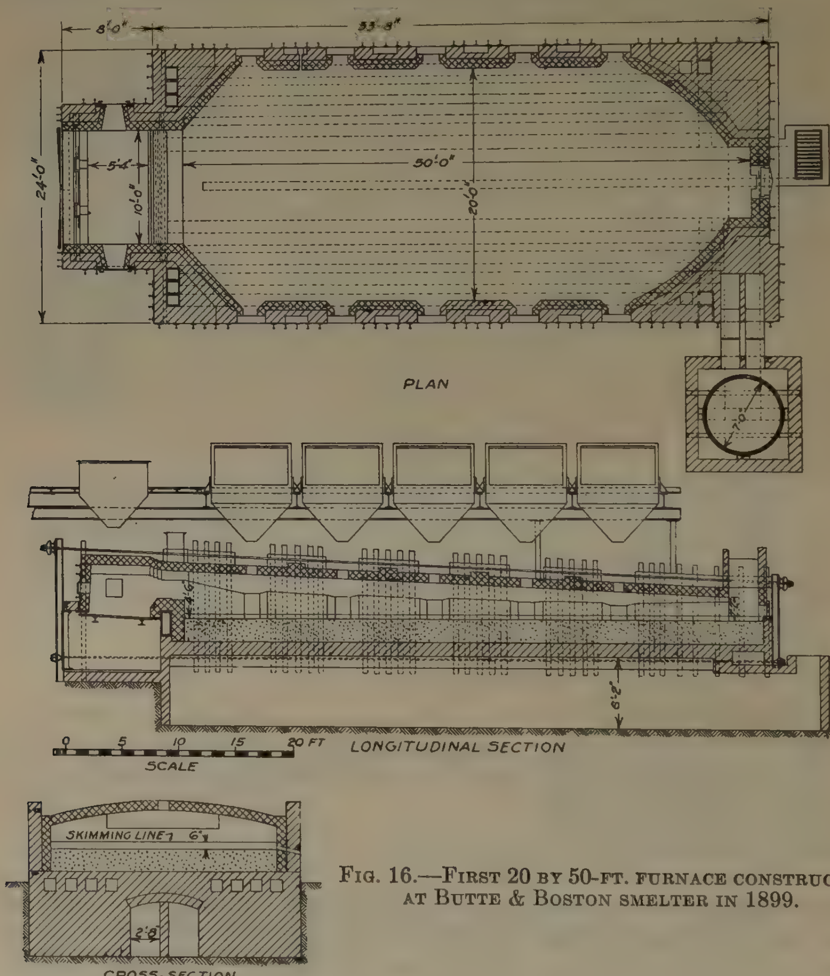


Fig. 16.—First 20 by 50-ft. furnace constructed at Butte & Boston smelter in 1899.

of 46 tons. The Lower Works furnaces were operated on Sand Coulee coal for about two years, from 1894 to 1896, without, however, achieving satisfactory results. The air was preheated to some extent by circulation under the furnace, and steam and forced blast were used under the grates, but performances were very poor. The amount smelted was only about 20 tons per 24 hr. per furnace with a ratio of 2 to 1. The ratio was not so bad, considering the low heat value of the coal, about 9500 B.t.u.,

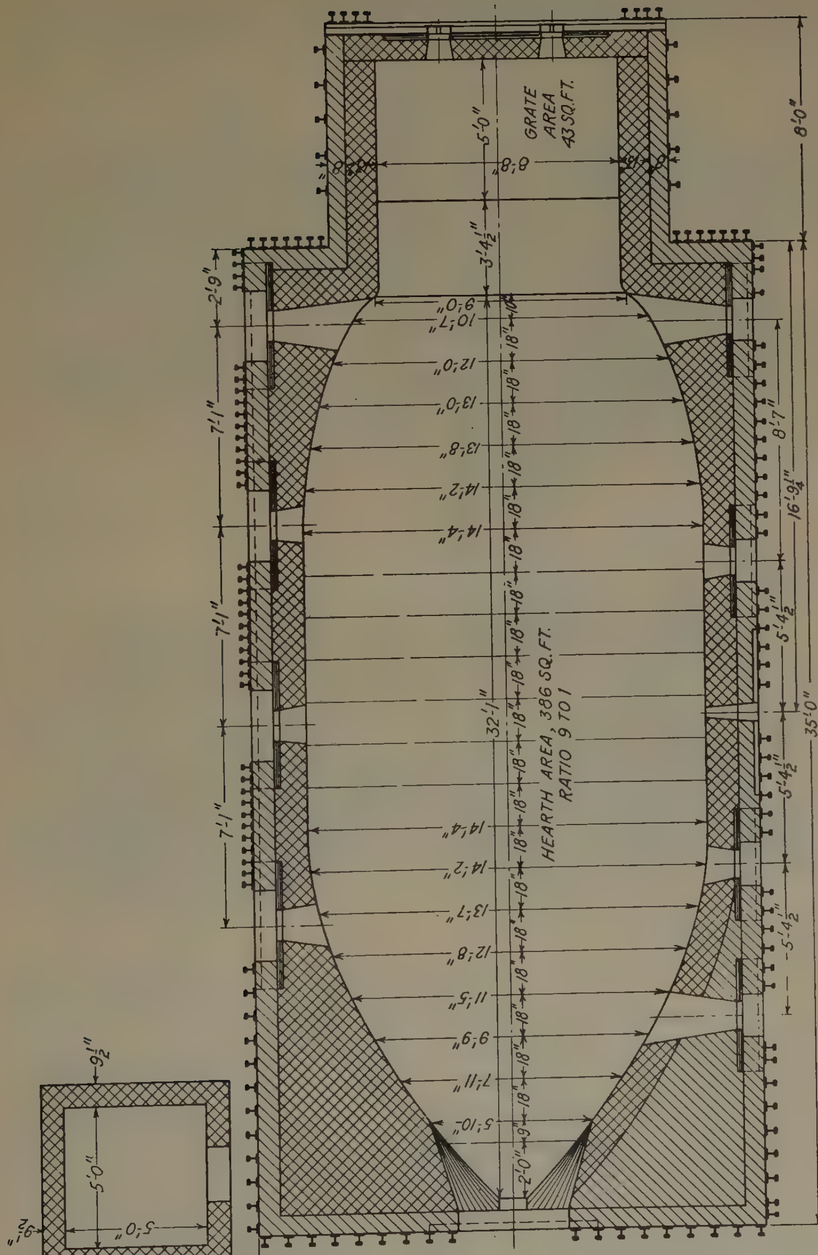


FIG. 17.—TYPICAL PLAN OF FIVE REVERBERATORY FURNACES AT THE LOWER WORKS, CONSTRUCTED ABOUT 1898.

and the high ash—from 20 to 25 per cent—but it was almost impossible to keep the furnaces hot and interruptions were frequent.

Roasting

The ratio of iron and sulfur in the concentrates from Butte ore has always, until recent years, been such that from 75 to 80 per cent of the sulfur had to be eliminated by roasting in order to make a matte of acceptable grade. It is outside of the scope of this paper to discuss

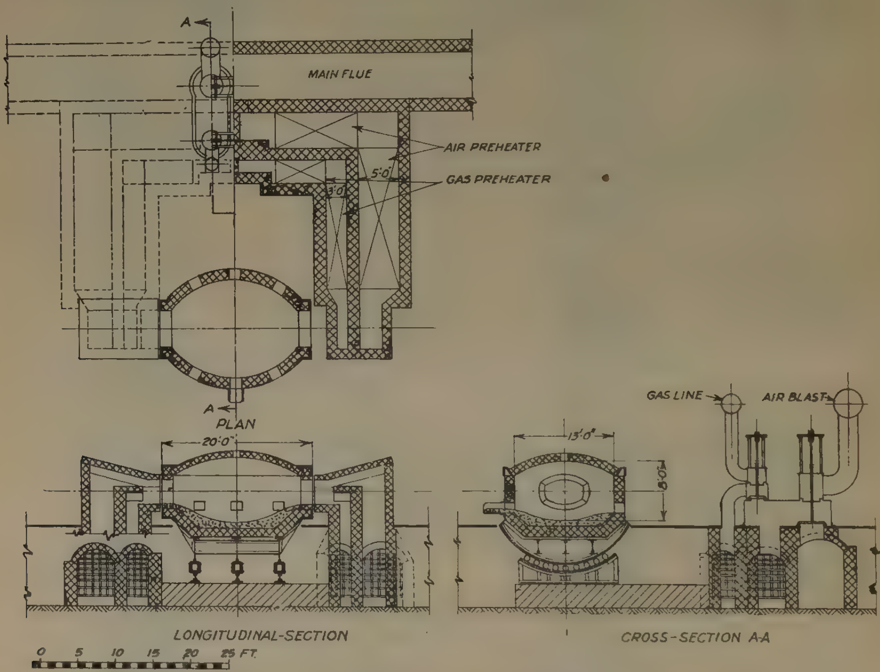


Fig. 18.—Early Great Falls gas-fired regenerative tilting furnace of 1894.

Montana roasting practice in detail but a brief résumé of developments in this field will not be out of place. The earliest roasters were rabbled by hand but it was not long before mechanical furnaces were adopted. For many years the Bruckner cylinder was generally used at Anaconda and Great Falls and, although it had the disadvantage of intermittent operation, it held its own against mechanically rabbled, straight-line furnaces, which were favored at Butte. Of the latter, there were many different designs, such as the O'Hara, Weathey, Ropp, etc. All of these furnaces required considerable fuel and maintenance costs were high.

Finally in the late nineties, the McDougall type of furnace began to attract attention. This furnace, as originally constructed, was small

and there was no provision for cooling the cast-iron shaft and rabble arms. Moreover, the rabbles and arms were cast in one piece. The Herreshoff furnace was a modified McDougall in which air cooling had been applied

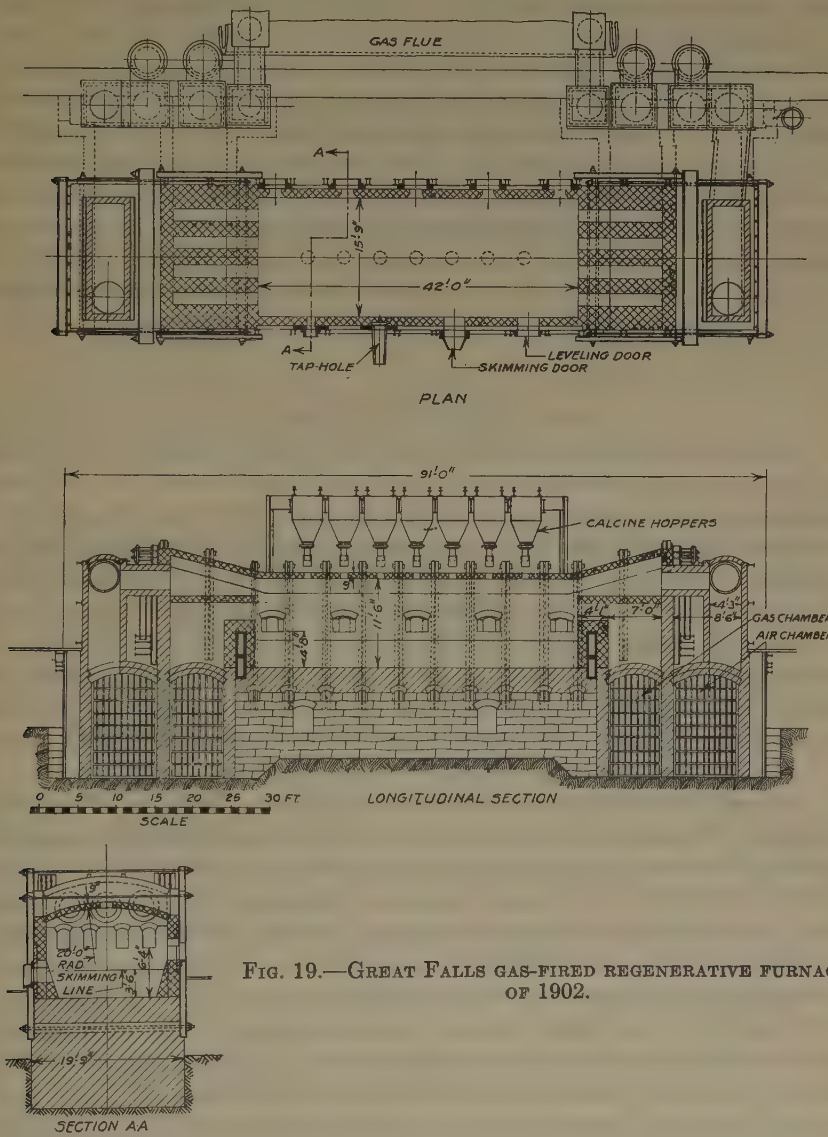


FIG. 19.—GREAT FALLS GAS-FIRED REGENERATIVE FURNACE OF 1902.

to shaft and arms but it too was available only in comparatively small units. These and other drawbacks were overcome in the Evans-Klepetko McDougall roasting furnace, which was designed and built at Great Falls and won instant favor. The six superimposed hearths were enclosed in a steel shell, 16 ft. in diameter, and the shaft and arms were

water-cooled. The rabbles were separate castings which slipped over flanges on the sides of the arms. Each of these furnaces roasted about 40 tons of concentrates per 24 hr. and used no fuel whatever. Maintenance expenses were moderate and roasting costs dropped to unprecedentedly low figures.

The multiple-hearth roasting furnace is now used everywhere, although the design of Evans and Klepetko has undergone some modification. The furnaces have been further enlarged and air cooling is generally preferred to water cooling. This, however, is due to the fact that in most localities the concentrates requiring roasting are deficient in sulfur. In Montana, on the contrary, the concentrates contained so much sulfur and developed so much heat that the charge sintered on the hearths and interfered with the smooth running of the rabbling mechanism unless the temperature was held down by the use of water, at least in the arms.

Tilting Furnaces

While the foregoing developments were taking place at Butte and Anaconda, a radically different type of smelting furnace had been constructed at the Great Falls plant of the Boston & Montana Consolidated Copper & Silver Mining Co. Ground for this plant was broken in the spring of 1890 and reverberatory smelting was started in April, 1892. The reverberatories were patterned after open-hearth steel furnaces of the tilting type, were equipped with regenerative chambers and were heated with producer gas. According to Hofman, the original furnaces, of which there were eight, had hearths 13 by 16 ft., but the earliest drawings in the files at Great Falls are some pencil sketches dated May 4, 1894, which show the hearth dimensions as 13 by 20 feet.

According to a memorandum in the office files, three out of the eight furnaces were operated by direct firing of coal during the last four months of 1892, the remaining furnaces being operated on producer gas with regenerative heating of the air and gas. The direct-fired furnaces averaged only 8.9 tons per day as compared with 25.2 tons for the gas-fired regenerative furnaces. The fuel consumption, on the other hand, was no better for the regenerative furnaces than for the direct-fired, being 1.12 tons of coal per ton of charge for the former and 1.13 for the latter. It is only fair to say, however, that lump coal was used in the direct-fired furnaces, whereas run of mine coal was used in the gas producers and under the boilers furnishing steam for them. The fuel cost in the gas-fired furnaces was $1.55 per ton of charge as compared with $2.81 for the direct-fired.

It appears that considerable trouble was experienced in operating the tilting furnaces. The regenerator checkers were too small and could not be cleaned without shutting down the furnace. The furnace roofs were

low and arched in one direction only and smelted out rapidly. There was considerable leakage between furnace and checkers. As a result of studies made in the fall of 1894, these defects were corrected and the hearth was apparently shortened to 16 ft., the length given by Hofman. These changes appear to have had the desired effect, as in 1895 the furnaces were averaging about 42 tons per day and were using only 0.77 tons of coal per ton of charge, corresponding to a ratio of 1.3 to 1. The calcine, which was charged hot, ran 20 per cent copper and 8 per cent sulfur, and produced a matte running from 50 to 55 per cent copper. No slag analyses have been found but such records as exist appear to indicate that trouble was experienced with removing the slag and that matte losses were such as to give serious concern. This led eventually to the abandonment of the tilting furnaces.

Stationary Furnaces Fired with Producer Gas

In 1896 No. 8 furnace was rebuilt as a stationary furnace with a hearth 30 ft. long. This was a marked improvement and smelted about 93 tons per day. In 1897, No. 7 furnace was built with a hearth 42 ft. 6 in. by 15 ft. 9 in. About the same time, the gas producers came in for attention and a new type capable of gasifying about 12 tons of coal, as compared with 6, was designed and installed. In 1900 and 1901 two more furnaces, Nos. 5 and 6, of the same hearth dimensions as No. 7 but of improved design, were erected and the old tilting furnaces 1, 2, 3, 4 and 5 were dismantled. Finally the last reverberatory, No. 4, was constructed in 1902. Later Nos. 7 and 8, which were of inferior design and construction, were dismantled, and the performance data that follow apply to the furnaces that remained, which were the last ones to be built; viz., Nos. 4, 5 and 6.

Generally speaking, it may be said that these furnaces smelted about 200 tons of charge per day with a coal consumption of 50 per cent; in other words, a charge to fuel ratio of 2 to 1. It is possible that these results could have been bettered had the furnaces been nearer to the producers and had the area of gas and air checkers been larger. The heating area was 2300 sq. ft. for the gas and 5800 sq. ft. for the air checkers at each end of the furnace. One of the largest items of operating expense was repairing these checkers as they became clogged with dust. Since this happened more frequently than repairs to the furnace, there was a loss of time and heat which was chargeable to the regenerative system itself and led to its ultimate abandonment.

The gas required per furnace was determined by tests to have been 14,300 cu. ft. per minute at a temperature of 790° F. and a pressure of 679 mm. A producer gasifying 12.5 tons of coal per 24 hr. yielded 2122 cu. ft. of gas per minute at this temperature and pressure. Under standard conditions of temperature and pressure, this gas had a calorific

value of 142 B.t.u. per cubic foot. Calculation further shows that, including the sensible heat in the gas delivered to the furnace, the heat in one pound of coal was distributed as follows: 71.5 per cent in gas delivered to the furnace, 21.9 per cent lost in conversion and 6.6 per cent lost in producer ash. From 20 to 22 producers were required for the operation of the three furnaces.

Performance of Furnaces

Data as to the performance of furnaces operating at Butte and Anaconda during the 10 years from 1880 to 1890 are meager and are none too plentiful for the decade 1890 to 1900. Such operating records as were no doubt kept have been for the most part lost or destroyed. Hofman gives the fuel consumption of two wood-burning furnaces, with hearths 10 by 15 ft., at the Colusa smelter, as 10 cords per 24 hr. when smelting 12 tons of charge; in other words, one cord per 1.2 tons of ore smelted. Coal-burning furnaces of this period smelted from 15 to 18 tons with a ratio of about 2 of charge to 1 of fuel. It was not until the furnaces were substantially enlarged and calcines were charged hot that the furnace capacity increased beyond 18 tons per day and the fuel ratio showed material improvement. Peters states that about 1893 the general run of furnaces at Butte and Anaconda measured about 14 by 20 ft. and smelted about 30 tons of cold ore per 24 hr. with about 8 tons of coal, which would correspond to a ratio of 3.75 to 1. This statement is probably optimistic as regards fuel ratio in the light of the results contained in Tables 1 to 4, which were compiled from correspondence and operating records in the possession of the writer. H. C. Bellinger states that such fuel ratios were obtained with hot ore at the plant of the

Table 1.—*Comparative Statement of Reverberatory Furnace Work at Butte & Boston and Colorado Smelters*

Average for October, 1899

	Butte & Boston Smelter Furnaces				Colorado Smelter Furnaces		
	No. 1	No. 2	No. 3	No. 4	No. 1	No. 2	No. 3
Hearth area, sq. ft.	428.6	260.9	272.4	188.6	772.2	532.0	628.2
Dimensions of hearth, ft.					50 × 18	36 × 14	45 × 18
Firebox area, sq. ft.	49.4	49.9	47.1	30.9	36.8	36.8	36.8
Ratio of hearth to firebox	8.68:1	5.23:1	5.78:1	6.11:1	21.0:1	14.5:1	17.0:1
Daily average tonnage smelted	81.8	73.3	77.0	58.3	79.2	51.6	58.6
Daily average tons coal burned[a]	28.5	23.1	23.6	19.9	20.1	17.1	16.8
Weight per charge, tons					10.0	7.0	8.0
Ratio of ore to coal	2.87:1	3.16:1	3.26:1	2.93:1	3.95:1	3.07:1	3.48:1
Pounds smelted per sq. ft. hearth area	381.4	561.6	565.6	617.6	205.0	194.1	186.5

[a] Sweetwater slack coal was used at Butte & Boston smelter; Diamondville lump at Colorado smelter.

Montana Ore Purchasing Co., using 16 by 35-ft. furnaces which smelted from 50 to 70 tons of charge per 24 hours.

The data of Table 1 are abstracted from a letter addressed to Frank Klepetko by W. C. Thomas, Superintendent of the Butte & Boston Works. It is dated Oct. 14, 1899, and compares the performance of furnaces at the Colorado Works with those at the Butte & Boston. The letter contains also the information given in Table 2 and the slag analyses of Table 3. All figures in the three tables were furnished by Mr. Pearce, manager of the Colorado smelter.

Table 2.—*Average Performance of Colorado Furnaces*
First 8 Days, October, 1899

	Furnaces		
	No. 1	No. 2	No. 3
Tons ore smelted per 24 hr	78.4	52.8	60.0
Tons coal burned per 24 hr	19.6	16.4	16.0
Ratio ore smelted to coal burned	4.0:1	3.2:1	3.7:1

Note: Charge consisted of 80 per cent hot calcine, 20 per cent cold ore. Copper contents of calcine, 8 per cent. Cold ores were mainly siliceous gold ores containing no copper. Coal used was Diamondville lump. During September, using Sweetwater slack, the furnaces at the Butte & Boston Works smelted 3.2 tons charge per ton coal.

Table 3.—*Colorado Smelter Slags, October* 14, 1899

Description of Sample	Analysis, Per Cent					
	Cu	SiO_2	FeO	Al_2O_3	CaO	ZnO
Thomas' sample, Oct. 7	0.4	33.8	37.0	9.3	2.0	16.0
No. 1 reverberatory, Oct. 9	0.4	34.6	38.0	10.0	tr.	15.5
No. 3 reverberatory, Oct. 9	0.5	35.4	38.0	9.9	tr.	14.0
Butte & Boston slag, September average	0.8	34.6	51.0	11.7	0.7	1.0

In a letter to Mr. Klepetko, Mr. Thomas comments on the operation of the furnaces as follows:

The charge on No. 1 reverberatory is about 10 tons which is dropped after skimming, leveled and fired for two hours, actual time. Then another charge of 10 tons is dropped which is not touched. This is fired on for two or three hours, stirred if ready, then skimmed. The two charges are smelted in from 6 to 6½ hours, furnace is tapped every three or four days, tapping out about 40 tons of matte of 55 per cent to 60 per cent copper.

The same system is followed on the other furnaces, No. 2 having a charge of 7 tons and No. 3 of 8 tons. They are skimmed three or four times in twenty-four hours, about 3 pots of slag being obtained to each skimming on No. 1, 2 pots on Nos. 2 and 3.

The coal used is Diamondville lump which has been broken and screened, more accurately described as egg coal. It is sometimes mixed with Rock Springs. While using Diamondville alone, the doors on the ash pit are kept closed and a jet of live steam is blown in under the grates through a two-inch opening on each side of the ash pit. This virtually constitutes forced draft, although not of a pressure such as would be obtained from a blower. When using Rock Springs (Sweetwater) lump coal, the doors are kept open and and no steam is used, natural draft alone being depended upon.

The furnaces are so constructed that a current of air is drawn underneath the hearth where it is warmed, passing thence into a wind-box or air chamber over the bridge, from where it is admitted through checker holes into the furnace. This air chamber is controlled by a set of dampers which are opened or closed to admit more or less air as the fireman thinks necessary. By this arrangement Mr. Pearce believes that he obtains almost complete combustion in the furnace.

The flues last about three months and there are practically no repairs to the stack linings, as only very little flame goes into the stacks. The repairs on the furnaces are slight. One roof has been in constant operation for about three years. Every time a furnace is tapped out it is clayed up, thus keeping the lining in good condition all the time. The furnaces are ironed and buck-stayed better than any furnaces I ever saw, never getting out of shape.

The consumption of coal is very light as you will see by the enclosed report. They now have in course of construction a furnace much larger than any now in use at their works from which they expect a much larger tonnage.

Mr. Klepetko in replying to this letter doubts the accuracy of the copper contents of the Colorado slag samples and thinks they may be considerably cleaner than monthly averages under normal operating conditions.

Table 4 shows typical performances of reverberatory furnaces at Butte, Anaconda and Great Falls during the years 1900 to 1904, inclusive. Shortly thereafter, the old plants at Butte and Anaconda were closed and their ores were transferred to the new works at Anaconda.

The coal used at Anaconda was about the same as that used at the Colorado plant. The reason for the better fuel ratio obtained in the furnaces at the Colorado smelter than in the 50-ft. furnace at the Butte & Boston is not known. This furnace was built according to drawings that had been made at the Colorado plant and was the largest furnace in Butte. It was similar to the 50-ft. furnaces which were later constructed at the New Works at Anaconda. The analysis of the charge was practically the same and at both plants calcines made up about 92 per cent of it. At this time both the Butte & Boston and the Colorado furnaces were operated on a charge averaging about 10 per cent copper, 35 per cent insoluble, 40 per cent FeO and 9 per cent S, and made a matte running about 45 per cent copper. The slags assayed 0.75 per cent copper, 38 per cent SiO_2 and 50 per cent FeO. The cost of smelting at the Colorado Smelting Works was about \$2.60 before general expense, which added about \$0.20. The cost of coal was \$1.35, leaving \$1.25 for all other direct expense including charging, slag disposal, repairs, etc. At the Butte &

TABLE 4.—*Typical Performances of Reverberatory Furnaces at Butte, Anaconda and Great Falls,* 1900 *to* 1904, *Inclusive*

Name of Plant	Furnace No.	Approximate Hearth Area or Dimensions	Period	Daily Average Tons Smelted	Tons Smelted per Ton Coal	Remarks
Colorado Smelter	1	18 × 45 ft.	6 months, 1904	80.4	3.83	
	2	14 × 36 ft.	6 months, 1904	75.2	3.75	
	4	18 × 50 ft.	6 months, 1904	96.8	3.81	
Butte & Boston..	1	17 × 25 ft.	18 months, 1904–05	53.7	1.75	
	2	17 × 22 ft.	18 months, 1904–05	48.7	1.96	
	3	17 × 22 ft.	18 months, 1904–05	42.8	1.74	
	4	20 × 50 ft.	18 months, 1904–05	99.1	2.58	
Anaconda Lower Works........	1	390 sq. ft.	July, 1901	69.9	3.15	Hearths about 14 × 33 ft.
	2	395 sq. ft.	July, 1901	68.3	3.11	
	3	430 sq. ft.	July, 1901	69.2	3.03	
	5	395 sq. ft.	August, 1901	59.1	2.99	Under-grate blast.
	7	395 sq. ft.	July, 1901	71.0	3.38	Without boiler.
	7	395 sq. ft.	July, 1901	55.4	3.10	With boiler.
	7	395 sq. ft.	August, 1901	56.5	3.21	With boiler.
Great Falls......	3 Fcs.	Av. 15.7 × 42.5 ft.	6 months, 1904	192.9	1.98	Gas-fired regenerative.

COAL ANALYSES, PER CENT

Plant	Moisture	Volatile Matter	Fixed Carbon	Ash
Colorado Smelter..................	6.0	36.3	45.2	12.0
Butte & Boston....................	5.0	38.3	47.4	9.3
Great Falls.......................	6.1	23.8	50.0	20.1

Boston Works, fuel costs were much higher and for the period in question averaged $2.27 per ton of cuprous material smelted.

The Anaconda furnaces shown in Table 4 were remodeled furnaces situated at the Lower Works. Two of the furnaces were equipped with boilers which were alternately connected and disconnected in order to ascertain the effect on furnace performance. The effect of the boilers on fuel ratio was not marked but the smelting rate was decidedly lessened. The same might be said about No. 5 furnace, which was operated with under-grate blast. These experiments were preliminary to the installation of the reverberatory furnaces at the New Works.

W. M. Kelly gives the following interesting account of the developments at the Lower Works at Anaconda from 1898 until the plant shut down about 1902. It seems that no changes of note had been made in the construction of the furnaces until that year, when some of them were lengthened to 28 ft. and widened to 12½ ft. This increased smelting capacity to 36 tons per day and fuel ratio to 2.75 to 1. Diamondville

run-of-mine coal was used and slags containing 0.75 per cent copper were made with matte running 50 per cent copper. This was an improvement over previous slags, which contained in excess of 1.0 per cent of copper.

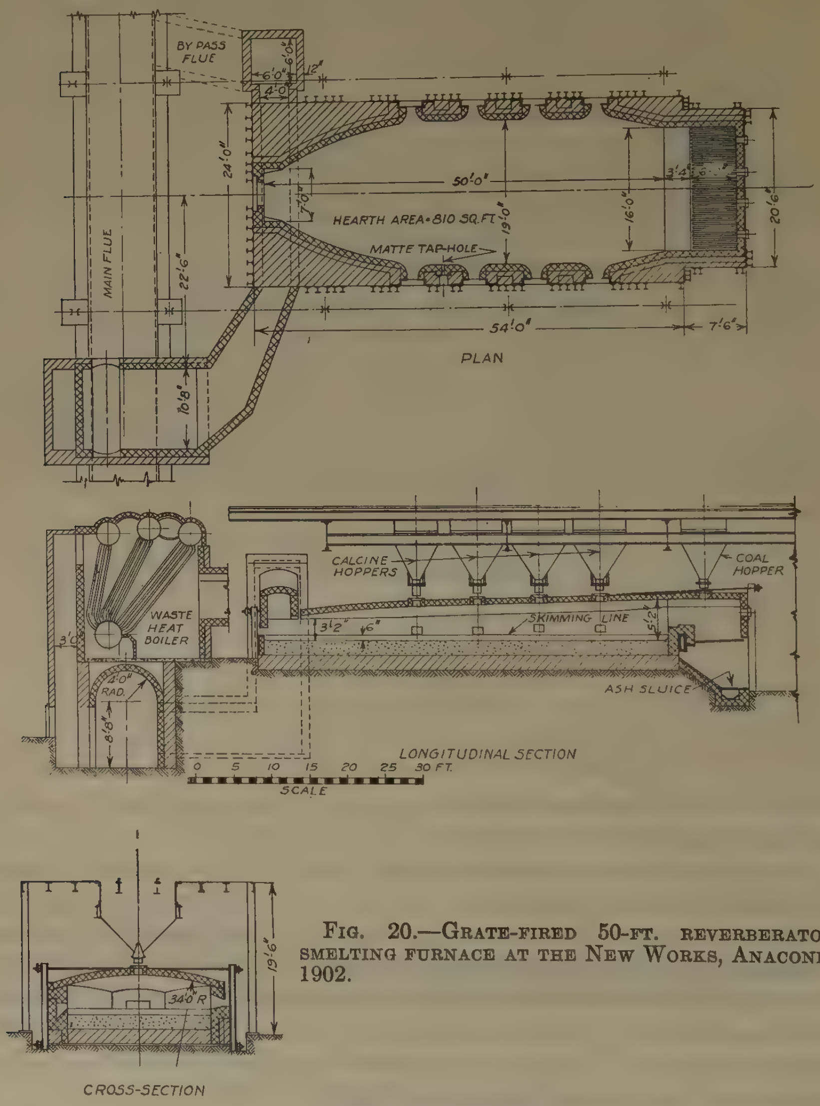


Fig. 20.—Grate-fired 50-ft. reverberatory smelting furnace at the New Works, Anaconda, 1902.

As a matter of fact some of the early slag dumps were subsequently reclaimed and used as blast-furnace flux at the New Works.

From the success attained in upper floor reverberatories, attention was next directed to the lower floor furnaces. The building of a 50-ft. furnace was discussed but it was finally decided to build five furnaces, 35 by 14 ft., and one 45 by 14 ft. The average capacity of these furnaces was 70 tons per 24 hr. with a ratio of 3 to 1.

There were five hoppers over each furnace and calcine was hauled from the roasters in mine cars. The charge was composed of calcine, roaster flue dust and lime rock.

In 1900, Mr. Cairns, who was assistant general manager, installed Stirling boilers on No. 5 and No. 7 furnaces. The fireboxes and flues were made smaller. The boiler tubes were baffled, leaving a very small opening from the boiler to the downtake. Doors were put on the ash pit of No. 5 and forced draft used at times. Results were

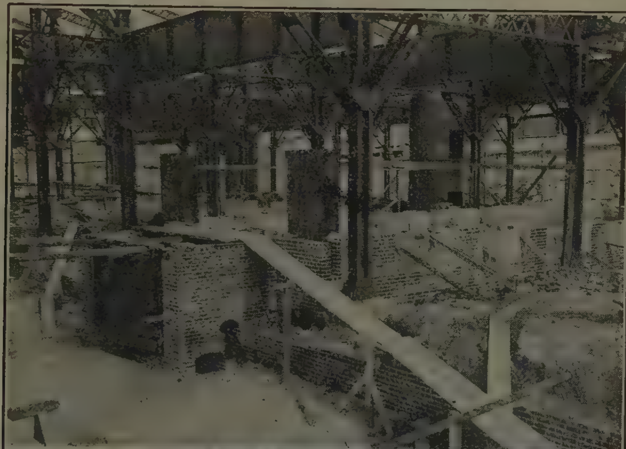

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FIG. 21.—FIREBOX AND BACK OF REVERBERATORY FURNACE UNDER CONSTRUCTION, 1901, NEW WORKS, ANACONDA.
FIG. 22.—DOWNTAKE AND FRONT OF REVERBERATORY FURNACE UNDER CONSTRUCTION, 1901.

unsatisfactory and later fireboxes and flues were rebuilt to their original size and the baffles were almost entirely removed from the boiler tubes. After this, fair tonnages were smelted but the results were not as good as were obtained without the boilers.

Before installing the Stirling boilers, an upright boiler had been tried in connection with No. 6 furnace on the upper floor. This proved a failure both as to furnace performance and generation of steam.

The Great Falls furnaces continued to operate as reported, without substantial change in design or performance, until they were closed down

in 1915. For nine months in 1911, for example, they averaged 183 tons per furnace day and smelted 1.9 tons of charge per ton of coal. While the tonnage and ratio were somewhat less than for the earlier period, the quality of the coal was also inferior, the ash having increased from 20 to 28 per cent and the B.t.u. having dropped from 10,860 to 9,382. Considering the inferior quality of the coal, it is apparent that the technique had improved somewhat. The grade of matte had been lowered from 46 per cent in 1904 to 32 per cent in 1911 and this had been accompanied by a drop in the copper content of the slag from 0.9 per cent in 1904 to 0.35 per cent in 1911.

FIG. 23.—REVERBERATORY FURNACE PLANT UNDER CONSTRUCTION AT NEW WORKS, ANACONDA, JUNE, 1901.

Nothing further need be said about the Great Falls gas-fired furnaces, as they were replaced by long furnaces, heated at first by large gas-producing fireboxes, each having an area of more than 400 sq. ft., and later with coal dust, following Anaconda practice of about 1915. These furnaces, of which there were two, had some structural peculiarities which are shown in one of the illustrations. The walls were incased in heavy sheet steel and the arch was kept from spreading by thrust posts instead of by tierods. The roofs were practically horizontal throughout their length and the opening into the flue was greatly enlarged. The furnaces operated satisfactorily but the steel casing added nothing to their performance and was expensive. No more furnaces of this type were built and after the ending of the war in 1918 copper smelting was all consolidated at Anaconda.

Fig. 24.—Washoe smelter, Anaconda, 1902. Old Works in background.

Anaconda Reduction Works

In 1902, the management of the Anaconda Copper Mining Co., at that time a subsidiary of the Amalgamated Copper Co., which had been formed to take over some of the copper mines and smelting plants of Butte, decided to abandon the works to the north of the city of Anaconda and consolidate smelting operations in a new plant which had been constructed on the south side of the valley. This plant, designed and built by Frank Klepetko and C. H. Repath, became known as the Anaconda Reduction Works and was enlarged from time to time until

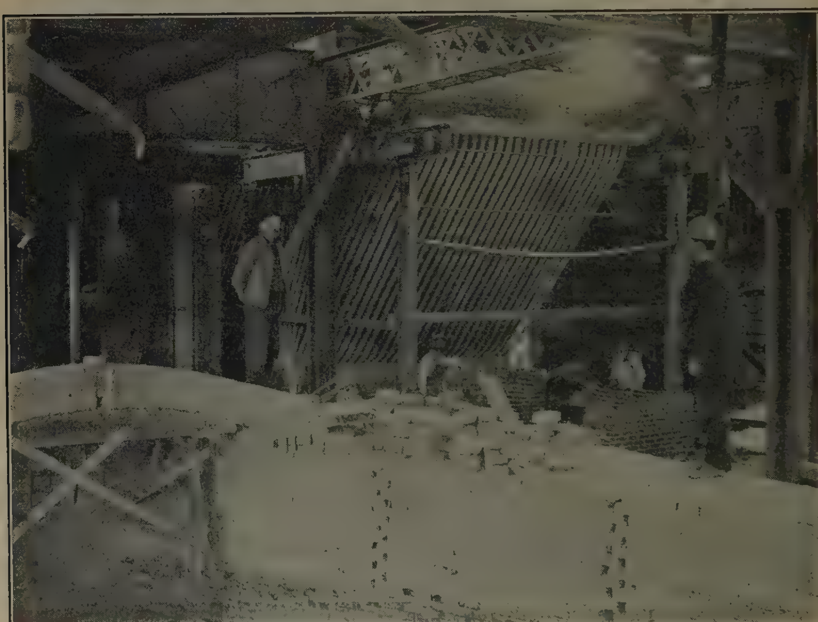


Fig. 25.—Stirling boiler on No. 1 furnace, 1904.

ultimately it treated all of the ores produced in the Butte district, with the consequent cessation of copper smelting at Butte and Great Falls.

The reverberatory smelting department of this plant as originally constructed contained fourteen 20 by 50-ft. furnaces equipped with undergrate forced blast. At first there were no waste-heat boilers, but by the end of 1902 all except No. 5 had been provided with one boiler to each furnace. There were two buildings, one with six and the other with eight furnaces arranged in two rows, with the fireboxes on opposite sides of a central aisle running lengthwise of the building and at right angles to the longitudinal axes of the furnaces. Overhead charge tracks entered

each building through the south side and each track passed lengthwise over two furnaces. It was this arrangement that made the subsequent construction of the long furnaces a relatively simple and inexpensive matter.

The gases passed through the boilers into header flues which served three and four furnaces, respectively, and led to a brick-lined steel stack,

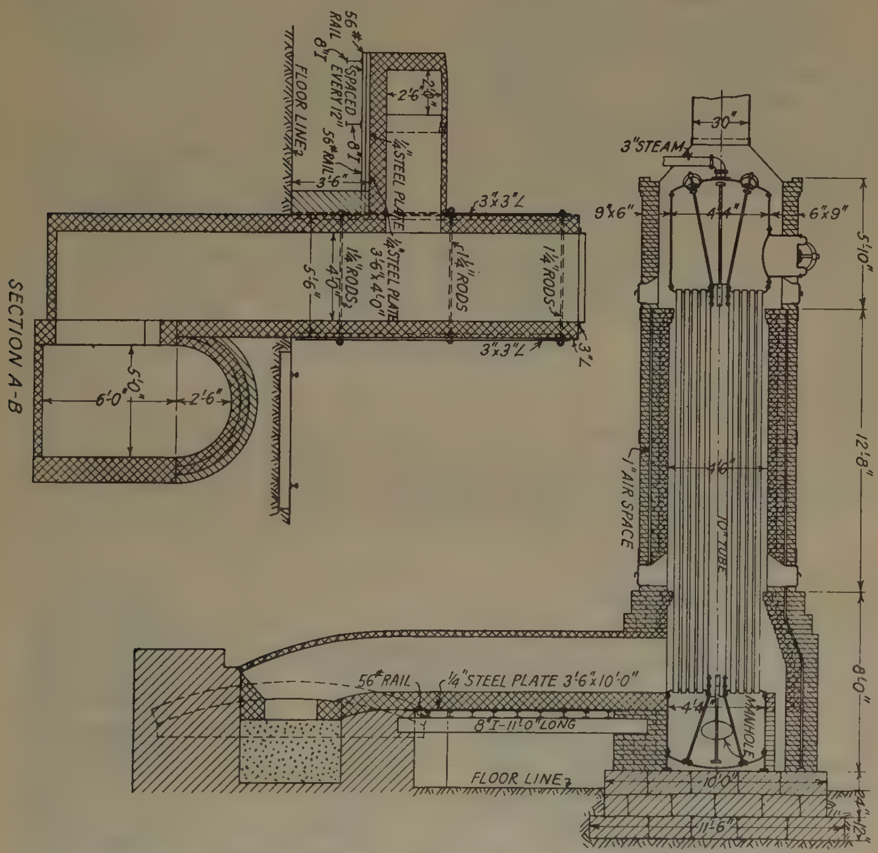


FIG. 26. —FIRST ATTEMPT TO RECOVER WASTE HEAT FROM REVERBERATORY-FURNACE GASES BY MEANS OF A BOILER. THIS CONSTRUCTION WAS NOT SUCCESSFUL.

20 ft. in diameter by 200 ft. high erected between the two buildings. Later a comprehensive flue and dust chamber system was constructed to handle the gases from all of the various departments of the works and the reverberatory stack was bypassed and dismantled. Coal and charge were brought in cars on the overhead tracks and dropped, as needed, from hoppers situated over the fireboxes and furnaces. Ashes and slag were sluiced away by water, the main launders discharging both at a substantial elevation above the valley floor, so that rehandling did not

become necessary for many years. The matte was tapped through holes in the side walls into ladle cars running on tracks at a lower level and delivered molten to the converters. Track scales were located at convenient points so that everything entering and leaving the furnaces except ashes and slag could be weighed.

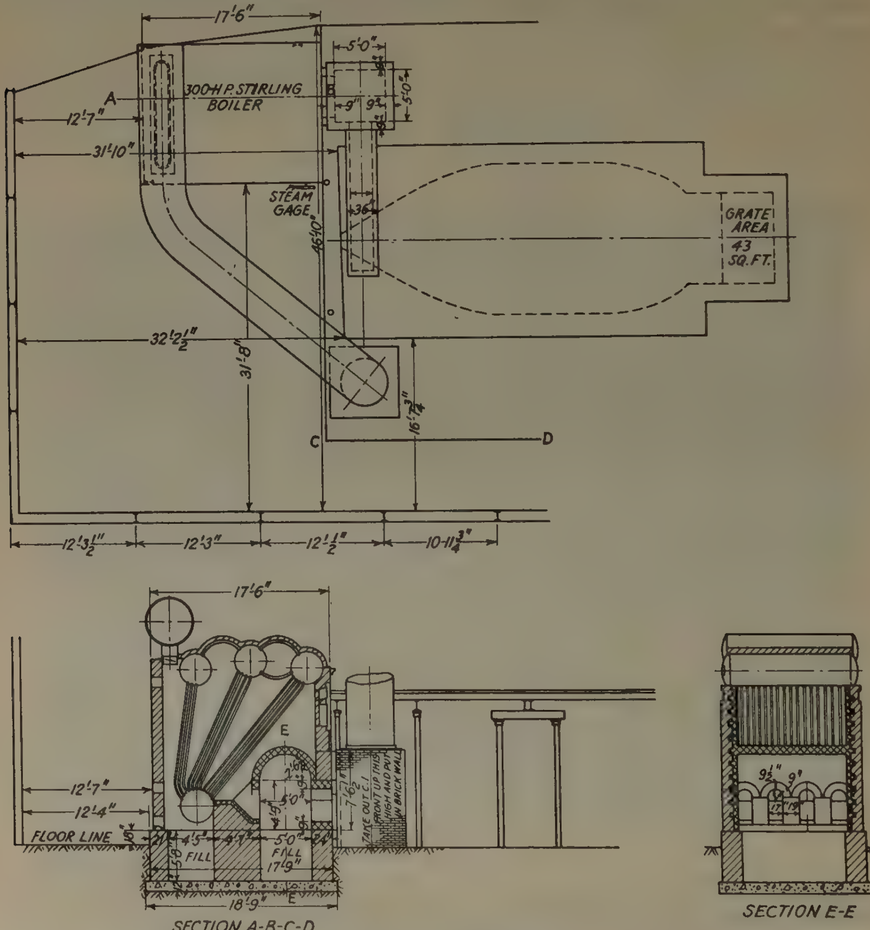


FIG. 27.—WASTE-HEAT BOILER ON NO. 7 REVERBERATORY AT LOWER WORKS, ANACONDA, ABOUT 1900.

This was the first successful attempt to generate steam from the gases leaving a copper-matting furnace.

The performance of these furnaces when forced blast was used was erratic and unsatisfactory. Table 5, compiled from some of the early records, compares two six-months periods, during one of which forced under-grate blast was used and during the other natural draft was used. After the furnaces were put on natural draft, both better and more uniform results were obtained.

TABLE 5.—*Comparison between Forced and Natural Draft on 50-ft. Furnaces, Anaconda Works*

	6 Months, 1902, July to December, Forced Blast	6 Months, 1903, January to June, Natural Draft
Furnace days operated	2310	1867
Materials smelted, tons:		
Hot calcines	213,617	217,721
Cold flue dust	9,329	12,751
Miscellaneous	337	228
Total	223,286	230,701
Coal burned, tons	96,905	82,804
Charge per furnace day, tons	96.75	123.56
Charge per ton of coal, tons	2.31	2.79

ANALYSES OF MATERIALS TREATED AND PRODUCED DURING THIS PERIOD, PER CENT

Material	Copper	SiO_2	FeO	CaO	S
Calcines	10.2	35.0	40.1	1.0	7.80
Lime rock		7.1	0.9	48.6	
Slag	0.50	40.4	44.8	3.1	
Matte	46.0		31.4		23.3

The coal contained approximately: 6.0 per cent moisture, 38.0 per cent volatile, 47.0 per cent fixed carbon, 10.0 per cent ash, and 11,000 B.t.u. wet or 12,000 dry.

DEVELOPMENT OF LONG FURNACES

In the latter part of 1903 it was decided to try the effect of lengthening one of the furnaces to 60 ft. This furnace did so well that another one was lengthened to 85 ft. The results being again satisfactory, other furnaces were lengthened to 102 ft., 112 ft., and 116 ft., respectively. At the same time each of the long furnaces was equipped with two boilers arranged in tandem. This work was practically all done during the years of 1904 to 1906, inclusive. I cannot do better than to quote from a paper by E. P. Mathewson and William Wraith. Table 6 shows the results obtained for furnaces of various lengths, over periods ranging from seven months to four years.

During January of 1905 a heat balance covering about four days (101 hr.) was made on the 102-ft. furnace, which during the test period averaged 272 tons of charge per 24 hr. with a ratio of 4.37 tons per ton of coal, per ton of charge. The charge assayed 9 per cent Cu, 31.3 per cent Fe, 2.9 per cent CaO, 8.1 per cent S and 26.1 per cent SiO_2. Table 7 shows a summary of heat distribution.

This test shows that 21.18 per cent of the total heat entering the furnace was usefully absorbed in smelting the charge. No allowance

TABLE 6.—*Comparison of Furnaces of Different Lengths*

Hearth Areas of Furnaces, Ft.	Tonnage per 24 Hours	Ratios of Cuprous Material to Coal	Averages
19 × 50	121.74	2.75	For 1 year
19 × 60	190.7	3.94	For 7 months
19 × 85	234.1	4.13	For 7 months
19 × 102	264.9	4.31	For 4 years
19 × 102	267.1	4.30	For 4 years
19 × 116	270.1	4.19	For 4 years

COPPER ASSAYS OF SLAGS FROM THE DIFFERENT FURNACES

Furnace No.	Length of Hearth, Ft.	Period Covered by Record	Per Cent Cu
1	50	Sept., 1903–March, 1904	0.50
6	60	March–Sept., 1904	0.44
1	85	May–Dec., 1904	0.42
1	102	Jan., 1906–May, 1911	0.39
4	112	July, 1906–May, 1911	0.38
1	116	Jan., 1906–May, 1911	0.36

ANALYSIS OF MATERIALS ENTERING AND LEAVING FURNACE

	Charge	Slag	Matte
Copper, per cent	9.31	0.39	41.7
Silver, oz. per ton	6.43	0.19	28.4
Gold, oz. per ton	0.028	0.0006	0.130
SiO_2, per cent	28.6	39.7	0.3
Fe + Mn, per cent	26.5	32.8	26.5
CaO, per cent	3.5	4.6	
S, per cent	7.3	0.9	25.8
Al_2O_3, per cent	5.2	7.0	Nil

Typical Diamondville coal was used, having the following analysis: 6.13 per cent moisture, 36.28 per cent volatile matter, 45.22 per cent fixed carbon, 12.17 per cent ash, 11,710 B.t.u. wet, 12,390 B.t.u. dry.

was made for heat absorbed or evolved in breaking up components of the charge and recombining them to form slag and matte, as it was not known whether the net result would be endothermic or exothermic but it was thought that the quantity of heat involved in either case was small. The fact that the calcine contains 5.08 per cent of the heat debited to the furnace and that only 18.92 per cent appear in slag and matte indicates that 26.8 per cent more coal would have been required if the calcine had been cooled to atmospheric temperature before charging. The benefit from charging hot calcine was obviously greater still in the preceding furnaces, which smelted only about 2½ tons of charge per ton

of coal. In these furnaces the usefully applied heat was only about 12 per cent of the total and with so low a smelting efficiency the fuel required to smelt cold charge would have been increased by 42 per cent.

TABLE 7.—*Summary of Heat Distribution in 102-ft. Furnace*
101 HR. IN JANUARY, 1905

Debit	B.t.u.	Percentage of Total Debited Heat
Net calorific value of coal *consumed*[a]	5,520,200,000	94.92
Heat in calcine and flue dust	295,500,000	5.08
	5,815,700,000	100.00
Credit		
Heat in slag	915,400,000	15.75
Heat in matte	184,700,000	3.18
Heat used to decompose limestone	131,000,000	2.26
Heat lost by radiation	674,900,000	11.55
Heat in air for cooling bridge plate	9,020,000	0.16
Heat in droppings from grate	45,330,000	0.78
Heat absorbed by boilers	1,908,380,000	32.81
Heat in escaping gases	766,800,000	13.18
Accounted for:	4,667,530,000	79.66

[a] The partly burned coal lost through the grates had a heat value of 766,800,000 B.t.u., or 12.2 per cent of the total, and is not included.

In general the method of operating these long furnaces was much the same as the method of operating the shorter furnaces which they displaced. The bottom was built up of sand and smelted in as usual, although the custom of smelting it in two layers was given up. On account of the size of the furnaces and the tendency to cut down on the number of doors, it became necessary to form the hearth once and for all and then smelt it in and season it with small charges of matte and slag, after which normal smelting commenced. The quantity of matte on the hearth was about 150 tons. It was soon found that all of the charge could be dropped near the bridge wall and that enough of it floated on the bath to the front of the furnace to take up the heat. The weight of a charge was 15 tons, of which 10 tons was dropped from a hopper nearest the bridge wall and the remainder from a hopper a little farther down. Fettling was done in the old manner once a month. The quartz, which contained over 99 per cent SiO_2, was crushed through a 3/8-in. screen and was thrown against the corroded areas, which were chiefly along the bridge wall and for a distance of 30 ft. or so down the side walls.

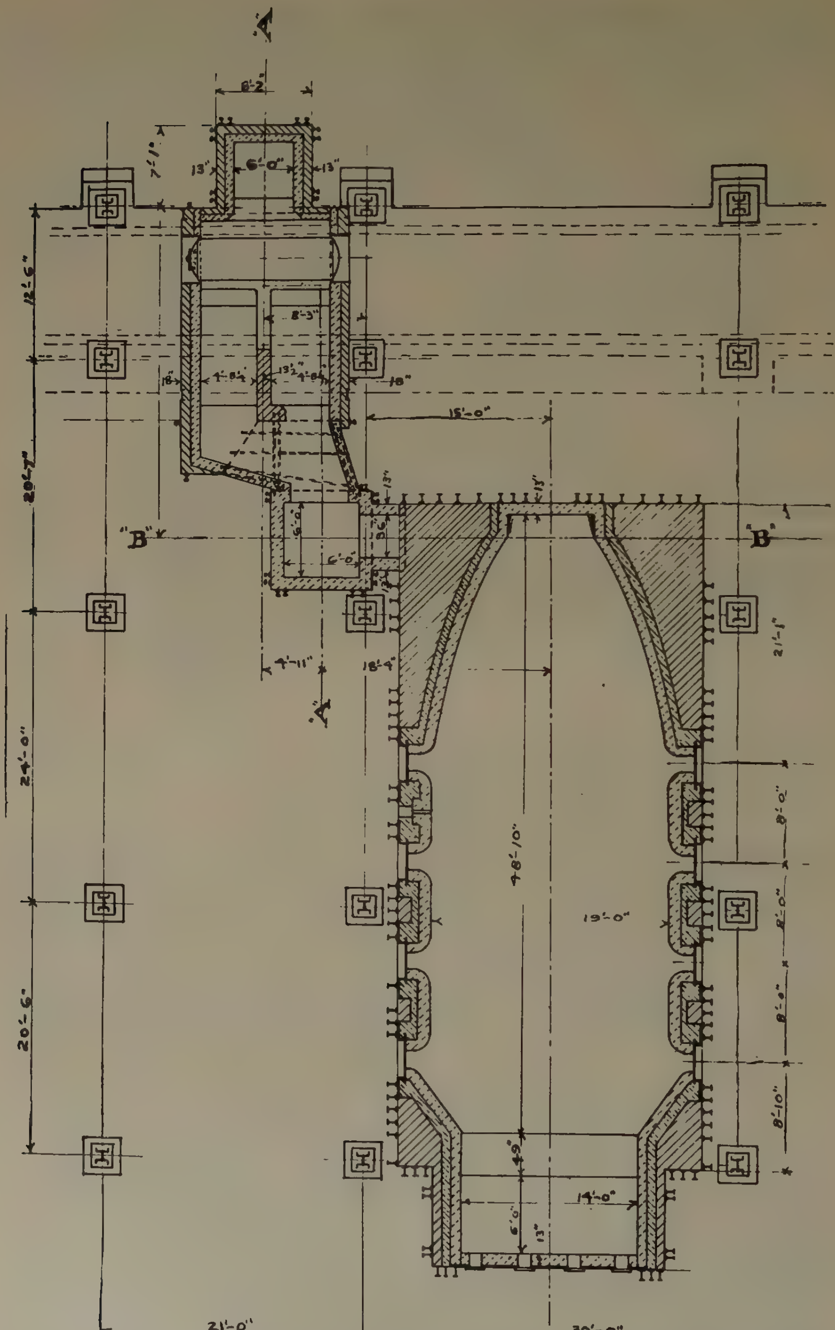
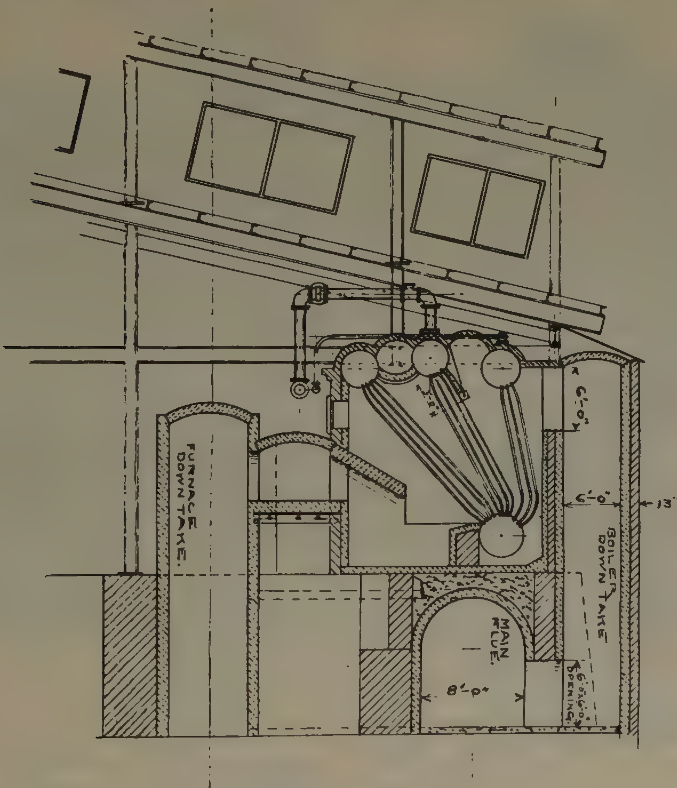


FIG. 28.—ANACONDA 50-FT. REVERBERATORY FURNACE OF 1903, SHOWING WASTE-HEAT BOILERS AND FLUE CONNECTIONS.

When the furnaces were lengthened, the fireboxes were enlarged from 6 by 10 or 14 ft. to 7 by 16 ft., or an increase in area from 60 or 84 sq. ft. to 112 sq. ft. The relation between the firebox and hearth areas, therefore, remains substantially unchanged, but in the case of the long furnace the heat is more effectively applied and much more intense heat development is obtained in the smelting region. The long



SECTION "A-A"

FIG. 28.—(*Continued*).

furnaces may be looked upon as consisting of: (1) a smelting zone extending about 50 ft. from the firebox, in which more than 80 per cent of the smelting was done, and (2) a zone for absorption of heat from the partially spent gases, in which the remainder of the smelting was done, and in which separation of matte from slag took place. In the settling area near the front, which was kept hot by the spent gases on their way out of the furnace, very little smelting went on and the remaining matte particles had an excellent chance to settle out of the slag.

This division of the functions of the furnace made for more nearly continuous operation than was previously possible. Charging was done

about six times during each shift of eight hours. Skimming was done at the beginning and about the middle of the shift. Matte was tapped whenever it was required by the converters. Slag and ashes were sluiced

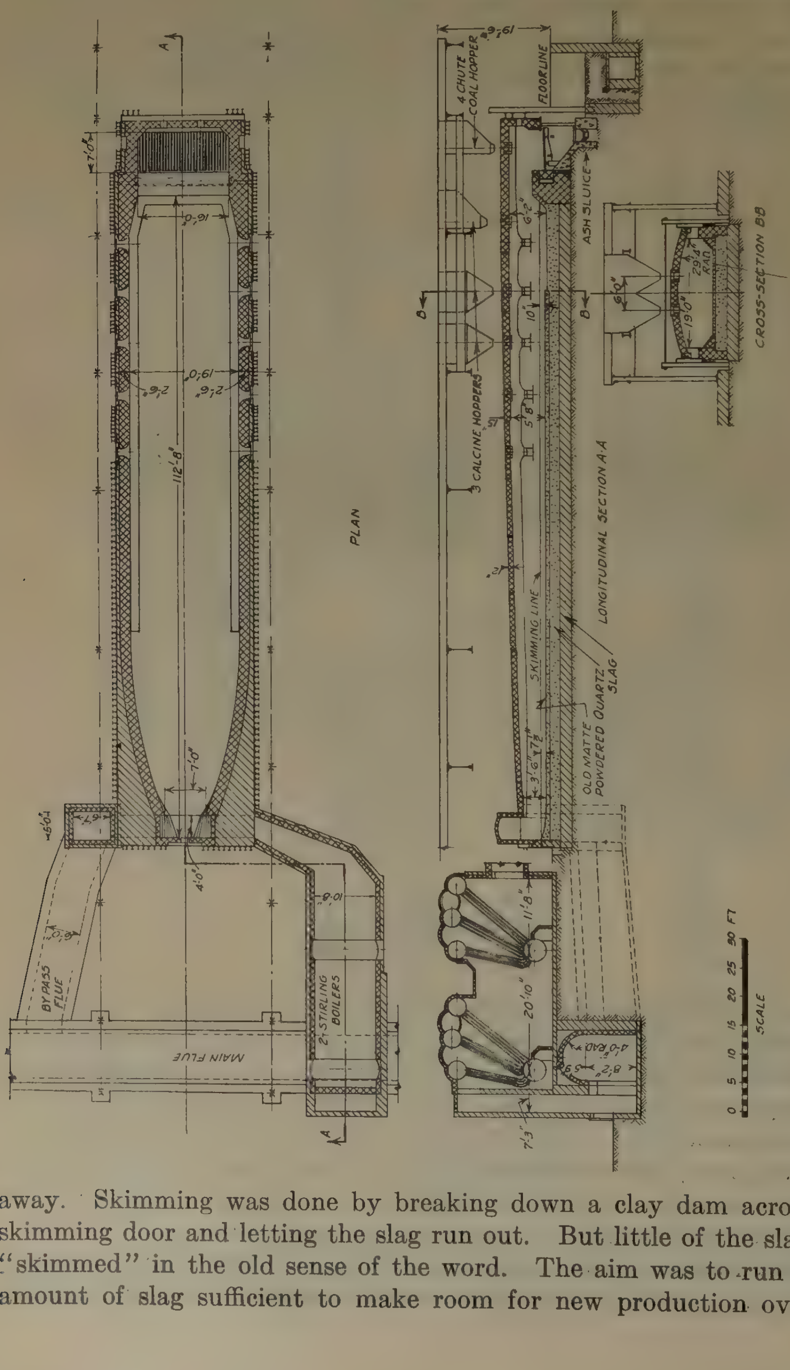


Fig. 29.—Grate-fired 112-ft. reverberatory smelting furnace constructed at Anaconda in 1906. Waste-heat boilers in tandem.

away. Skimming was done by breaking down a clay dam across the skimming door and letting the slag run out. But little of the slag was "skimmed" in the old sense of the word. The aim was to run off an amount of slag sufficient to make room for new production over the

next four hours. Only once a month, when the furnace had to be emptied of its matte to permit of fettling, was "skimming" done in the old sense of removing all of the slag from the matte.

The labor required for operating one of the long furnaces was practically the same as for a 50-ft. furnace, and since the long furnace smelted more than twice as much per day, there was a considerable saving in labor costs. Repairs were kept down by constructing the roof of 20-in. silica bricks over the smelting region. This part of the roof lasted about

Fig. 30.—First 100-ft. reverberatory furnace, Washoe smelter, Anaconda, July 25, 1904, showing No. 1 furnace from front.

nine months, the rest of the furnace about five years, the bottom practically indefinitely. Coal dropping through the grates was saved in jigs and the small pieces of coke were used in the blast furnaces as fuel with moderate success. Grating was done twice a shift and resulted in the loss of much of the fuel bed in the firebox and a great deal of the heat in the furnace. It was this that made the maintenance of the matte and slag pool so important, as the heat accumulated in the molten bath quickly restored normal conditions once the fresh coal started burning again.

Oil-fired Furnaces

The results obtained at Anaconda with the long furnaces attracted immediate attention throughout the smelting districts of the West and similar furnaces were constructed in Utah, Arizona, Nevada and at Cananea, Mexico. Most of these furnaces used oil for fuel. In the

Cananea furnace coal dust was first tried with indifferent success, after which oil was substituted. The results obtained with oil were, as might have been expected, much better than with run-of-mine coal burned in fireboxes and suggested that similar results might be obtained by the use of pulverized coal. Three hundred tons per furnace day was about as much as could be smelted under the best conditions, in a furnace heated with run-of-mine coal burned in a firebox, as it was impossible to obtain a temperature of much more than about 2400° F. in the smelting zone. The rate of smelting is determined by the difference between the temperature at which the charge fuses (which may be estimated at, say, 2200° F.) and the temperature in the smelting chamber. This difference in a grate-fired furnace, therefore, is only 200°, whereas with oil or coal dust a differential of 300° or 400° can easily be maintained, besides which the time and heat losses resulting from periodical grating are eliminated.

Oil-burning furnaces smelted regularly between 500 and 600 tons of charge per furnace day with about two-thirds of a barrel of oil per ton of charge. This corresponds to 3,600,000 B.t.u. per ton of charge as compared with 5,600,000 B.t.u. when using coarse coal at Anaconda. It was apparent that much better efficiencies might be expected from the use of pulverized fuel, but after the abandonment of pulverized coal at Cananea it remained for David H. Brown, metallurgist for the Canadian Copper Co. at Copper Cliff, Ontario, to demonstrate that it could be applied successfully. The abandonment of coal-dust firing at Cananea was due primarily to the fact that oil became available at low cost, but it appears also that there was some difficulty with coal dust because the grinding had not been adequate. The pulverizers used failed to grind all of the coal sufficiently fine, with the result that coarse particles settled out on the slag bath and formed what was referred to as an "ash or coke blanket," which tended to slow up smelting and gave trouble during skimming.

Coal-dust Firing

About the middle of 1913, an investigation was made of the furnaces at Copper Cliff, as a result of which No. 8 reverberatory at Anaconda was altered to burn pulverized coal. Details relative to the grinding plant are given in a paper by L. V. Bender and it will suffice to say that Raymond mills with air separators were used and no trouble of any kind was caused by the settling of coarse particles on the bath. In remodeling this furnace, the firebox was included as part of the smelting chamber and the opening into the flue and the dimensions of the "verb" were greatly enlarged in order to allow for the much larger volume of hot gases to be generated in the furnace.

In the "grate-fired" furnace, it was necessary to have the roof slope rather steeply toward the skimming plate in order to bring the gases into

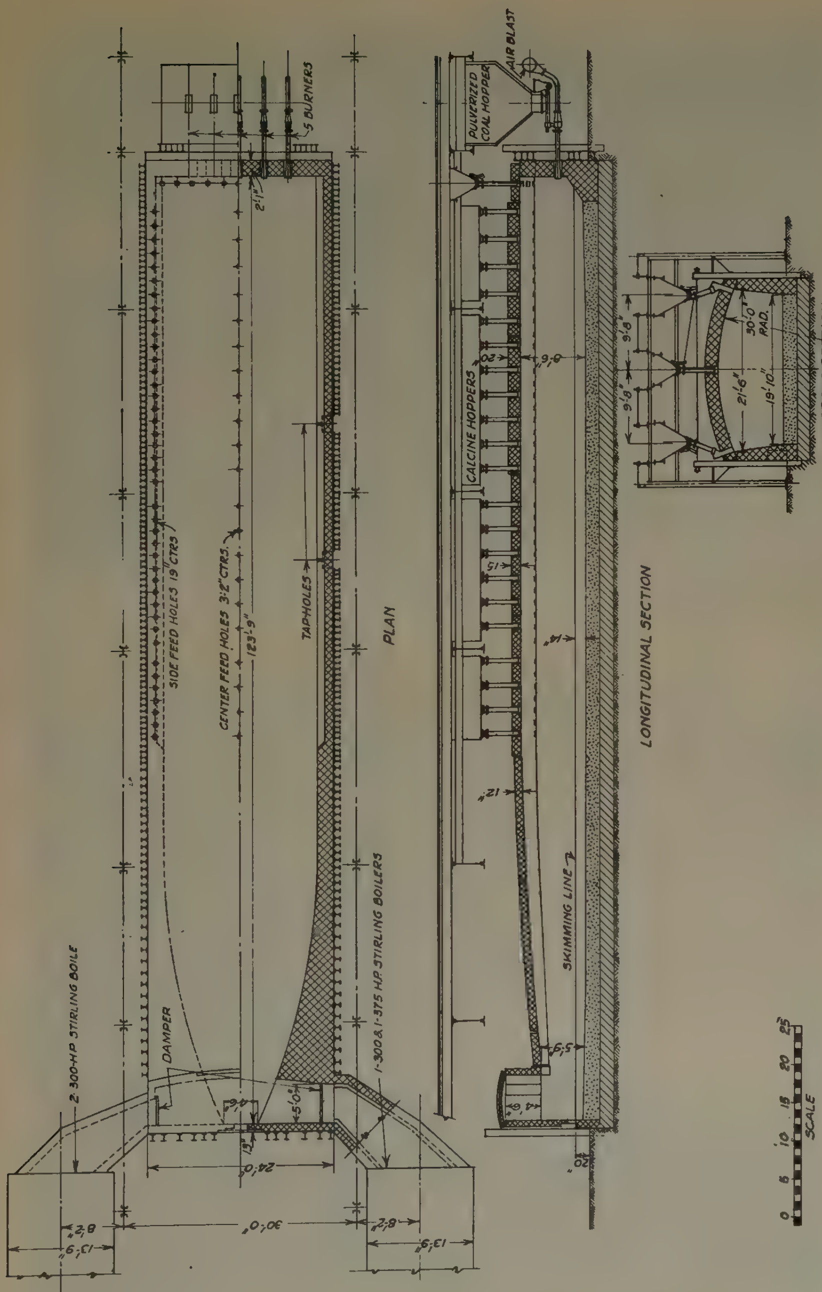


FIG. 31.—No. 8 FURNACE OF 1914. THIS WAS THE FIRST COAL-DUST FIRED, SIDE-CHARGED REVERBERATORY CONSTRUCTED AT ANACONDA.

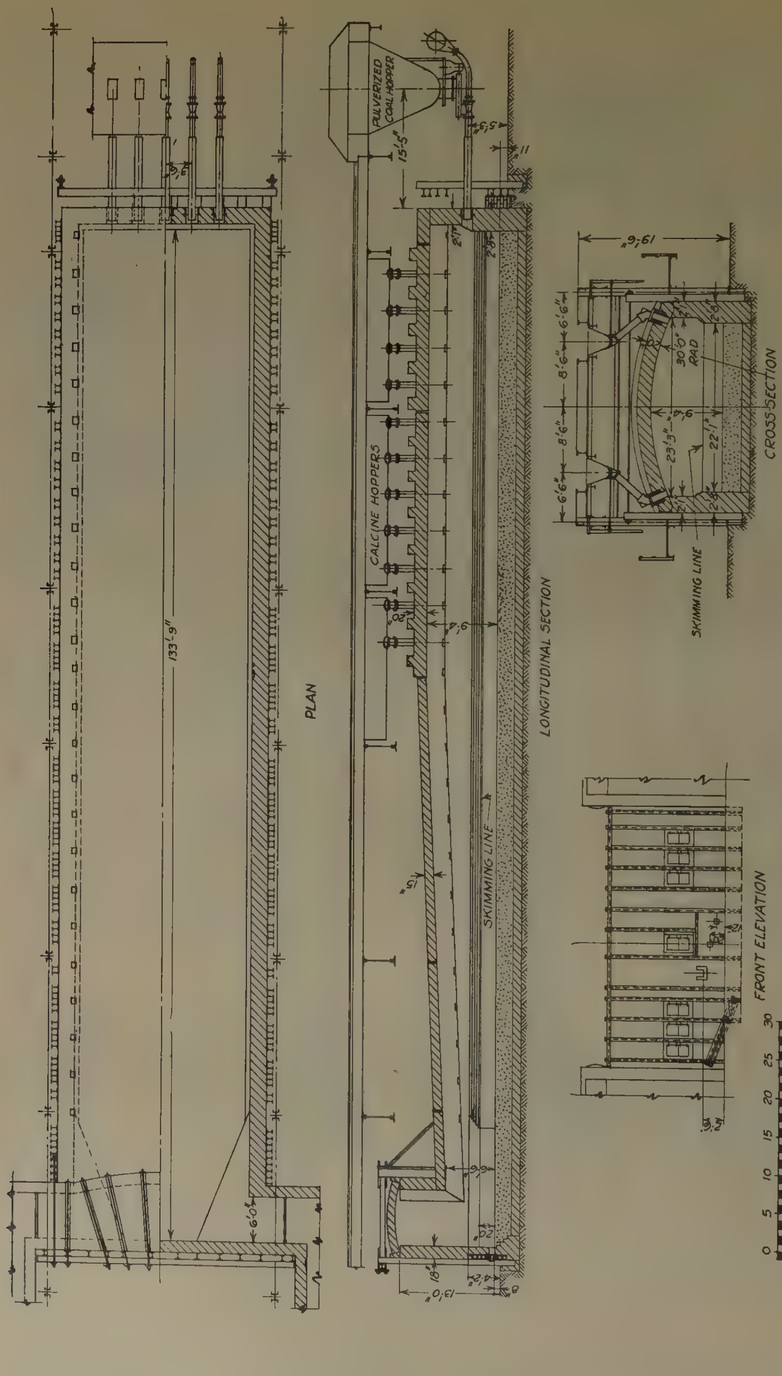


FIG. 32.—ANACONDA REVERBERATORY SMELTING FURNACE OF 1916 AND ABOUT 10 YEARS THEREAFTER.

FIG. 33.—COAL-DUST BURNERS AND BACK OF REVERBERATORY FURNACE No. 5 AT ANACONDA.

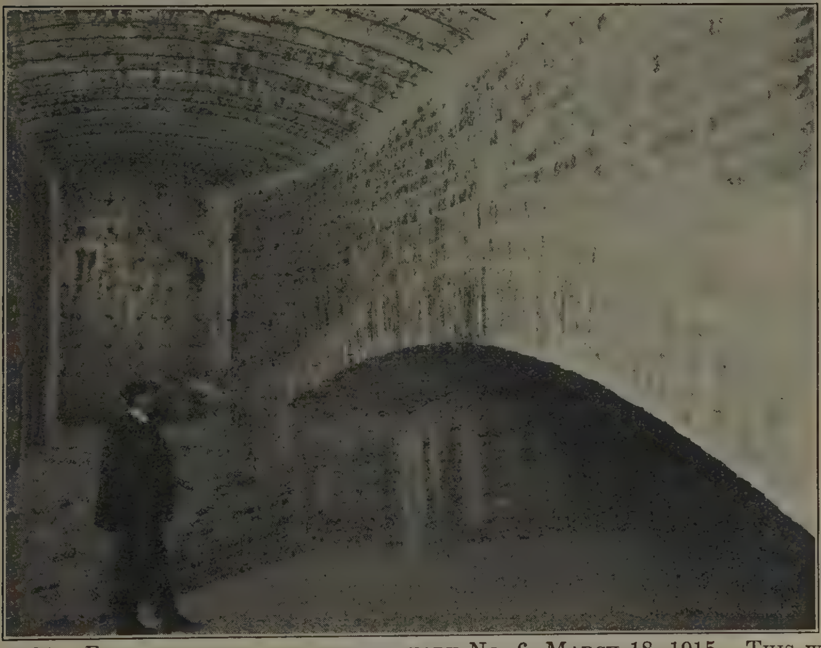


FIG. 34.—FLUE AND VERB OF REVERBERATORY No. 6, MARCH 18, 1915. THIS WAS ONE OF THE FIRST FURNACES RECONSTRUCTED TO BURN COAL DUST.

closer contact with the slag and keep it from freezing in the "skimming bay." In these furnaces the rate at which heat could be developed in the firebox was limited by the available draft and by the resistance offered by the coal bed, and generally there was not much margin. Consequently attention had to be paid to utilizing as effectively as possible the heat remaining in the gases at the front of the furnace. The slope of the roof and the drawing in of the walls were for the purpose of increasing the velocity of the gases, compensating for their contraction, due to cooling, and bringing them in closer contact with the bath. With coal dust so much heat could be developed that there was no longer any

Fig. 35.—Reverberatory and coal-pulverizing plant, Anaconda Reduction Works, 1917.

difficulty in keeping the front of the furnace hot but plenty of area had to be allowed to permit the gases to escape into the flue. Otherwise no draft could be maintained at the firing end of the furnace and an adequate volume of secondary air could not be drawn in.

At Copper Cliff, the practice had been adopted of dropping most of the charge along the side walls of the furnace, thus protecting these walls to some extent from the flames and slag. No. 8 furnace was equipped with hoppers along the side walls so that this method of charging could be used. It was successful but the bath, which had been considered so necessary in previous practice, quickly froze up as the charge crowded toward the center line of the furnace. There was much speculation at the time as to whether better results might have been obtained by staying with the old methods of charging and maintaining a pool of matte and slag throughout the length of the furnace but it was not until years

afterwards, during the Carson litigation, that the question was definitely answered in the negative, as will appear later. The steady development

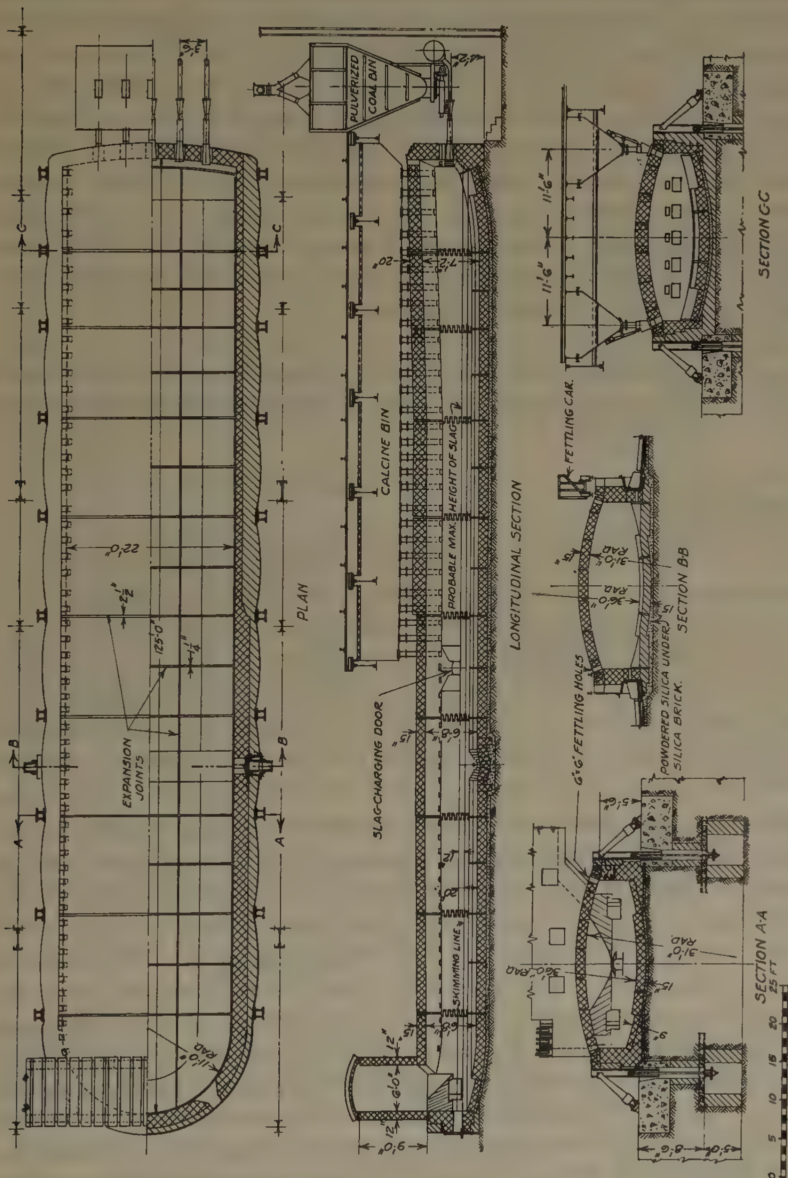


FIG. 36.—GREAT FALLS STEEL-CASED REVERBERATORY FURNACE OF 1915.

of heat and the high temperatures obtainable with coal dust or similar fuel, as compared with coarse coal burned on grates, make the deep bath unnecessary.

No. 8 furnace was about 20 ft. wide inside by 124 ft. long. The roof for a distance of about 94 ft. from the burners was horizontal and at the center line of the arch was 8 ft. 6 in. above the floor line. At the front the roof sloped to a height at the verb of 7 ft. 2 in. This furnace was so successful that the remaining seven furnaces were rebuilt to use coal-dust and side charging and at the same time were enlarged to a width of about 21 ft. and a length of 143 ft. Several years later, they were widened to 22 and 23 ft. and shortened to 134 ft., for structural reasons. Figures for width are variable, because these dimensions are inside and depend on the thickness of the walls. After some experience with the furnaces, charging thimbles and skimming plates were water-cooled. Matte was tapped at the front instead of at the side, as the side tap-holes were difficult to keep open when side-charging.

Table 8 compares the work of No. 7 grate-fired, and No. 8 coal-dust fired, operating side by side, the former on the old system, the latter on the new; also the performance of No. 7 furnace after being enlarged and fired with coal dust.

TABLE 8.—*Comparison of Grate Firing and Coal-dust Firing*

Furnace No.	Dimensions, Ft.	Total Tons Smelted	Tons per Furnace Day	Charge per Ton Coal, Tons	Length of Campaign, Days	Date of Starting
7	19 × 112	75,129	242	3.81[a]	310	May 5, 1914
8	20 × 124	116,823	465	6.19	251	July 1, 1914
7	21 × 143	71,962	579	6.49	124	July 8, 1916

[a] The performance of the furnaces had fallen off from earlier years owing to an increasing percentage of fines in run-of-mine coal from the company's mines at Diamondville.

A heat balance based on average results for the six months ending June 30, 1920, is given in Table 9. The percentage of total input usefully applied (i.e., sum of heat units in slag, matte and limestone) is 27.3 per cent, and this may be called the smelting efficiency of the furnace. The test made in 1905 gave a corresponding figure for the grate-fired furnace of that date of 21.2 per cent. The fuel ratio at that time was 4.37 to 1 as compared with 6.27 to 1 for the first half of 1920, using coal of substantially the same fuel value in both instances. These figures appear to indicate a smelting efficiency of 30.4 per cent for the coal-dust furnaces. In the 1905 calculations, however, the heat value of the carbon in the coal and clinker which dropped through the grates, and which had been determined at 12.2 per cent of all coal charged into the firebox, was deducted from the total heat value of coal charged. Making allowance for this, the true fuel ratio becomes 4.98 to 1, and from this we deduce a smelting efficiency, for the coal-dust furnaces of 26.7 per cent, which checks very well with the later results. In making this calcula-

tion the same heat values for slag and matte were used as for the earlier calculation in order that they might be comparable. The values used were 579 B.t.u. per pound for molten slag and 375 for molten matte.

TABLE 9.—*Heat Balance Based on Average Results*
For Six Months Ending June 30, 1920

	B.t.u. per Furnace Day	Per Cent
Input		
Coal	1,827,676,000	94.63
Heat in calcine	103,680,000	5.37
Total	1,931,356,000	100.00
Output		
Slag	335,820,000	17.40
Matte	96,000,000	4.97
Limestone (decomposition)	96,567,800	4.99
Boilers	579,406,800	30.00
Waste gases	347,644,100	17.94
Radiation, conduction and other losses	475,917,300	24.70
Total	1,941,356,000	100.00

In addition to the considerable improvement in smelting results obtained when using Diamondville coal, pulverizing enabled inferior coals to be used and this led to the adoption of the system at Great Falls, where Lochray and Belt coals were the only ones that could be had at a reasonable price. These coals could not be used economically on grates because it was impossible to develop a high enough temperature. Demond[1] gives the results of tests using Bear Creek coal. With natural draft, about 190 tons of charge were smelted per furnace day with a fuel ratio

TABLE 10.—*Results with Pulverized Low-grade Coal*

Coal	Tons Smelted per Furnace Day	Tons Smelted per Ton Coal	B.t.u. in Wet Coal	Moisture in Coal	Ash in Coal	B.t.u. per Ton Charge
Lochray	409.4	5.38	9,730	8.0	20.9	3,620,000
Bear Creek	406.7	5.57	10,580	9.0	12.7	3,800,000
Diamondville	475.8	7.24	12,470	5.6	8.1	3,450,000

of about 2.2 to 1; with forced draft (under-grate blast) about 225 tons with a fuel ratio of about 2.7 to 1. In a test using Belt coal the furnace froze up, although coal was being burned at the rate of 98 tons per day.

[1] C. D. Demond: Economy and Efficiency in Reverberatory Smelting. *Trans.* A.I.M.E. (1914) **49**, 735.

With the pulverized-coal system it not only became possible to use these coals but they did an amount of smelting that was very nearly proportional to their heat values, as is well shown in Table 10. The small differences in the heat units required can be accounted for by differences in the moisture and ash contents of the coals. An oil-fired furnace at McGill operating on a similar charge used about 3,600,000 B.t.u. per ton of charge, thus indicating about the same efficiency for coal dust as for oil.

While a larger percentage of the developed heat is usefully applied for smelting in pulverized-fuel furnaces than in grate-fired furnaces, the amount of steam generated in the waste-heat boilers is also greater, being about 650 boiler-hp. for No. 8 furnace after remodeling, as compared with about 500 boiler-hp. under the old system. The coal ash gave some trouble in the boilers but was kept under control by somewhat more frequent blowing of the tubes and cleaning out of ashes and clinker which settled in the space beneath.

When these furnaces were being operated on the old type of calcine, which contained only about 10 per cent copper, slag was made so rapidly that periodical skimming was no longer necessary and a steady stream ran from the furnace just as from the settler of a blast furnace. Thus the ideal of continuous operation in reverberatory practice was attained and the furnaces operated for months at a stretch without any attention except to see that they had an abundance of charge to work on and that the matte level did not get too high.

Improved Treatment of Slag

When the World War came to an end the blast furnaces at Anaconda were shut down. The flotation process and the improvements in reverberatory smelting combined to render them obsolete. The ore going to them had been siliceous and chilled converter slag was required for fluxing. With the blast furnaces down, two methods of treating the converter slag were investigated; one being the standard method of distributing it among the reverberatories, the other subjecting the slag to special treatment in a single furnace. The latter plan was adopted because the reverberatory furnaces at Anaconda are in a building some distance from the converters and at a higher level, and by erecting a furnace in one end of the converter plant much handling and transportation of molten slag, with consequent skull formation, was avoided. Recently conditions have changed again, so that the slag is now being distributed to the regular furnaces.

The performance of this furnace is described in a paper by Frederick Laist and H. J. Maguire.[2] The total treatment was about 1000 tons per day, of which 55 per cent was molten converter slag containing 4 per

[2] F. Laist and H. J. Maguire: Reverberatory Furnace for Treating Converter Slag at Anaconda. *Trans.* A.I.M.E. (1920) **64**, 585.

cent copper and 45 per cent was solid charge. The effluent slag assayed 34.1 per cent SiO_2, 53.3 per cent FeO, 1.7 per cent CaO and 0.55 per cent Cu. The matte contained 40 per cent copper. The total materials charged per ton of coal burned were 11.1 tons. The tons of solid charge smelted per ton of coal were 6.13. The precipitation of the copper from the converter slag was accomplished by the addition of concentrates high in sulfur. At the same time siliceous ore was banked along the side walls, so that the highly ferruginous converter slag would have ample opportunity to pick up silica on its way to the outlet, from which it flowed in a continuous stream at the rate of 800 tons per day. In order to give more time for contact, the furnace was made 153 ft. long—probably the longest reverberatory that has ever been built.

Value of Side Charging

About 1924 the copper-smelting companies were confronted with serious litigation involving the right to use the method of side charging. It is outside the scope of this paper to discuss the merits of the Carson case or of the decision which was finally rendered, but certain investigations that were made while the case was being tried are interesting in that they settled certain controversial points bearing on the mode of operating reverberatory furnaces. Assuming that the validity of the patent might be established, it was obviously desirable to determine in advance what the probable value might be of side charging as compared with charging in the old way. At first it was thought that the question involved was merely side charging versus center charging, or heaping the charge against the side walls versus piling it up away from the side walls. Company's Counsel, however, took the position that it might be necessary to go further back than that and actually reestablish conditions as they had been prior to the adoption of side charging and coal-dust firing, with the exception that coal dust would continue to be used for heating. In other words, the practice of the old grate-fired furnaces, with their deep baths extending throughout the length of the furnaces, was to be reestablished.

No. 7 reverberatory was chosen for this purpose, was equipped with hoppers for center charging and with side doors for fettling. Two other furnaces, Nos. 5 and 6, were also equipped for center charging but were operated without the continuous deep bath. No. 5 had water-cooled side walls while No. 6 had extra thick brick walls. As a comparative furnace, No. 4 was chosen and operated as a normal side-charged furnace. This work was not directly connected with the trial and was done purely for the purpose of ascertaining the facts; which, it may be stated here, were never used. It was realized that a short test would not be sufficiently accurate and that to even out irregularities comparative records should be kept for about a year. The same kind of material obviously

had to be fed to all of the furnaces and it was necessary to maintain not only like chemical composition but like proportions of hot and cold, coarse and fine, etc., and to see that all of the furnaces were kept amply supplied with charge at all times. It was also important to ascertain the amount of dust leaving each furnace, the slag losses, repair costs, operating cost, etc. Careful accounts of all these were kept and it may be said that very little difference in any of these factors was found. The advantage of the side-charged furnace lies mainly in the fact that it operates with

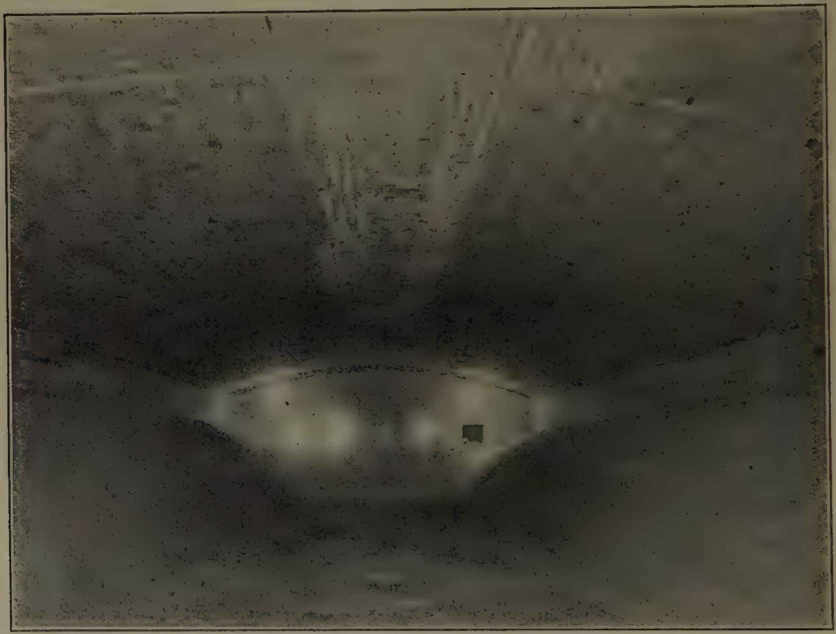


FIG. 37.—INTERIOR OF REVERBERATORY FURNACE NO. 7 LOOKING TOWARD TAPPING END.

less attention on a greater variety of feed, is more easily kept in condition and, because it contains no large body of matte and slag, is safer. This, however, is not a peculiarity of the side-charged furnace, as is shown by Nos. 5 and 6, which also contained no matte pool in the smelting zone.

Table 11 shows the results, which are remarkably alike and appear to demonstrate that it makes no material difference how a reverberatory furnace is charged or whether it operates with or without a pool of matte. The important thing is to maintain a constant evolution of heat and keep plenty of unsmelted charge in the furnace so that the flames and hot gases shall at all times have something to work on. These findings are at variance with the opinions held by smeltermen when the deep bath was considered so essential to good work (prior to 1913) and also with statements made when side charging first came into general use (about

1915), when the opinion was prevalent that side charging increased furnace capacity. An inspection of the figures in Table 11 shows that neither of these views was correct. Methods A, B, C and D were as follows:

Table 11.—*Comparison between Side and Center Charging and Dry Hearth versus Deep Bath*

Mode of Operation	Furnace Days	Tons Smelted	Tons per Furnace Day	Tons Smelted per Ton Coal	Copper in Slag, Per Cent	Dimensions of Furnace, Ft.
A	485	262,483	542	6.72	0.35	22 × 133
B	454	249,801	550	6.79	0.35	23 × 133
C	445	230,702	533	6.70	0.36	23 × 133
D	364	200,855	552	6.88	0.32	23 × 133

A.—This furnace, No. 4, was side-charged and operated in the regular manner with the so-called "dry hearth"; that is to say, a plateau of infusible accretions, chiefly magnetite, in the smelting zone, on which were shallow pools of matte and slag which overflowed into a fairly deep pool where settlement took place, and which occupied the front of the furnace for about 50 ft. back from the skimming plate. The pool contained about 50 tons of matte. The elevation of the plateau was by measurement several inches above the skimming-plate level, which makes it evident that no bath in the previously accepted sense of the word existed in the smelting region of this furnace.

B.—Furnace No. 5 was "center-charged"; that is to say, the charge was dropped on to the "dry hearth" from two rows of hoppers, which were nearer to the center line of the furnace than to the walls. There was no banking whatever of charge against the walls and these were kept from corroding through by water-cooled cast-iron plates set into the brickwork, which was thus protected from slag for a vertical height of about 2 ft. The charge was piled up on the hearth, which was kept in very much the same physical condition as the hearth in the side-charged furnace. In other words, hearth conditions in No. 5 were much the same as in No. 4 but the charge was piled in a double row down the middle instead of against the walls.

C.—This was No. 6 furnace, "center-charged" and "dry hearth." It was operated exactly like No. 5 but the walls were not water-cooled. Instead, the brickwork was made very thick where it was apt to come in contact with slag, the idea being to have enough thickness of brick to last out a campaign of about seven months. This, of course, reduced the width of the furnace at the slag line considerably (to 19 ft.), and may account for the fact that this furnace smelted at a somewhat slower rate than the others. However, the performance of the furnace was impaired less than might have been expected.

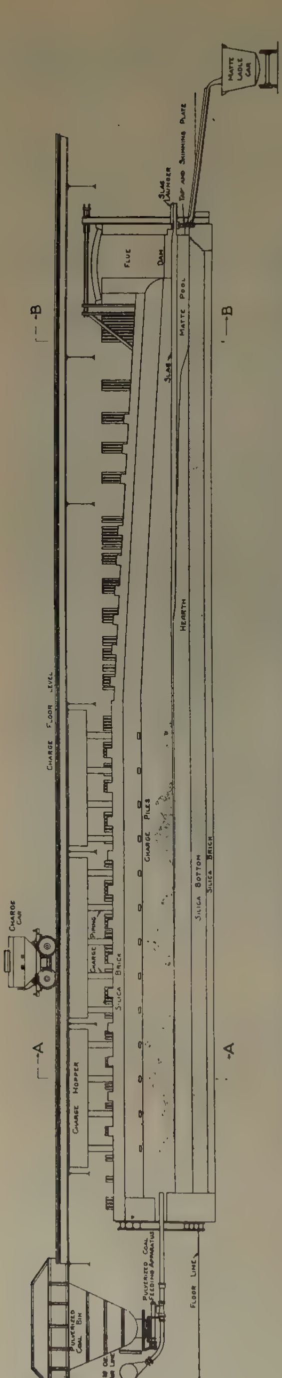
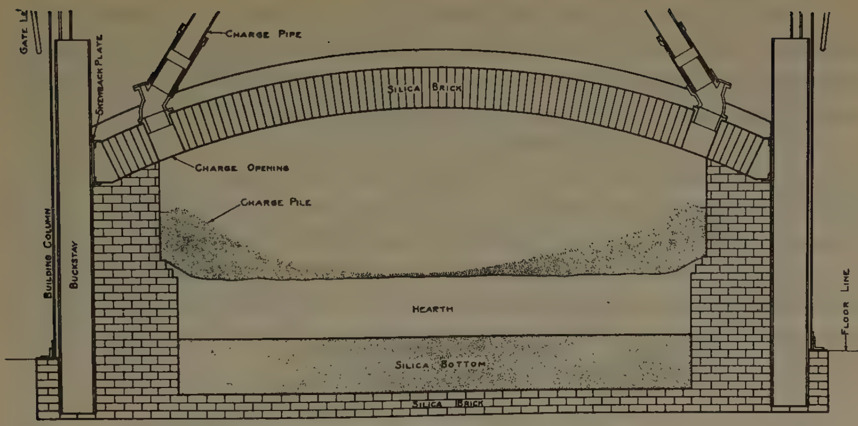


FIG. 38.—INTERIOR LONGITUDINAL VIEW OF SIDE-CHARGED REVERBERATORY OF ABOUT 1925. Shows "dry hearth" and diminutive matte and slag pool at front end.

D.—No. 7 furnace was run in the old style with a deep bath carefully maintained throughout the length of the furnace at a depth of about 2 ft. Charges were dropped at intervals of about ½ hr. in the smelting region, as was done during the days of grate firing. The more rapid smelting rate, of course, necessitated more frequent charging, since the quantity of charge dropped at one time remained about the same; viz., 10 to 15 tons. Smaller weights of charge were tried but seemed to make little if any difference. The walls were protected from corrosion by fine siliceous copper ore and coarse copper-bearing tailings from the Old Works, which were shoveled through openings cut for the purpose through the side walls and ordinarily kept closed by counter-weighted silica-slab doors. The furnace held about 200 tons of matte and may have been favored slightly by being fed somewhat less than its share of coarse material which had a tendency to sink to the bottom of the bath and stay there. However, it may be said that the composition of the charge to all four furnaces was substantially the same and that considerable care was taken to keep it so throughout the prolonged period of the test.

The figures in this tabulation speak for themselves. The tests extended over so long a time and are based on the smelting of so much material that there is every reason to believe that the results are essentially correct. They certainly indicate that the performance of a reverberatory is very slightly influenced by the mode of charging or the character or condition of the material on the hearth. What little difference there is in favor of the center-charged bath furnace may be explained by the slightly smaller percentage of cold coarse material in the charge and



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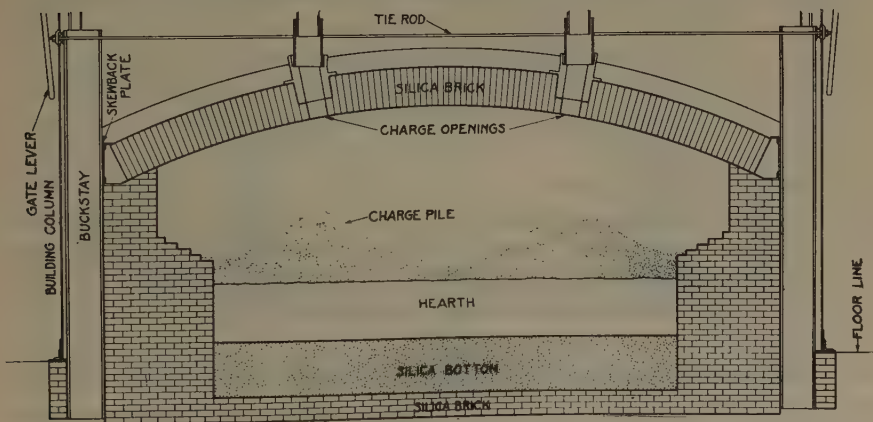

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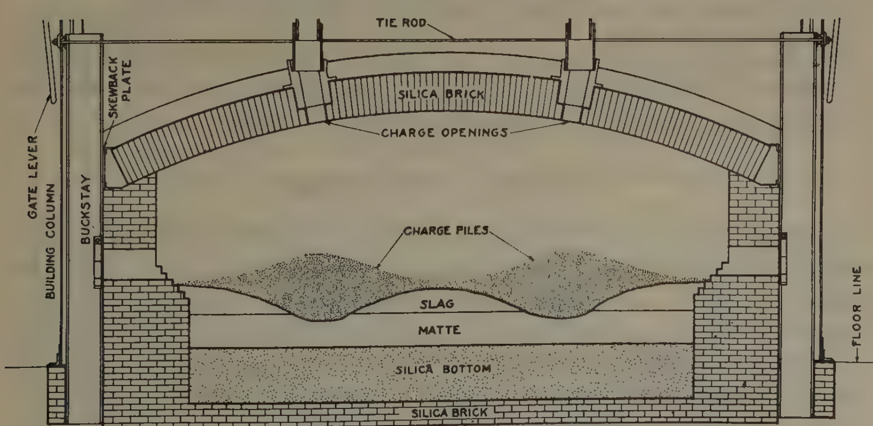

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FIG. 39.—CROSS-SECTION SHOWING "DRY HEARTH" AND CHARGE PILES RESTING ON SOLID "MAGNETITE" HEARTH.

FIG. 40.—CROSS-SECTION OF CENTER-CHARGED, "DRY HEARTH" SHOWING CHARGE RESTING ON SOLID "MAGNETITE" HEARTH.

FIG. 41.—CROSS-SECTION OF CENTER-CHARGED, "DEEP-BATH" FURNACE, SHOWING CHARGE FLOATING ON MOLTEN SLAG AND MATTE BATH.

the correspondingly larger percentage of hot calcine. The most notable difference here is the lower copper content of the slag, which was rather expected, but its value is scarcely sufficient to offset the greater operating difficulties and the hazard connected with holding so large a body of molten matte in the furnace.

For several years, the Anaconda reverberatories were operated as center-charged, the cutting of the slag and the danger of matte "break-outs" being obviated by the use of water-cooled cast-iron plates set in the brickwork of the side walls throughout the length of the furnace. These proved so satisfactory that they were retained as standard equipment even after side charging was resumed. Side charging had never

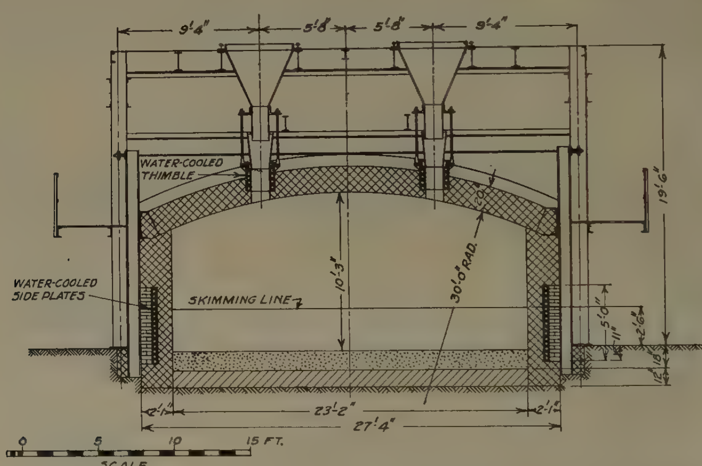

FIG. 42.—CROSS-SECTION OF ANACONDA CENTER-CHARGED, WATER-COOLED REVERBERATORY FURNACE OF 1928.

more than partly protected the walls and break-outs occurred at more or less frequent intervals. The water-cooled plates practically stopped this dangerous nuisance and helped to keep the furnaces in shape.

HANDLING A FINE CHARGE

The character of the charge underwent a change about 1927, when selective concentration was adopted, and became very much finer and higher in copper. The calcines from these concentrates were very hard to handle, made tremendous clouds of dust when dropped into the hoppers and would not form satisfactory embankments against the walls. In order to mitigate the transfer nuisance, special hoppers were constructed with large outlet pipes so that the calcines flowed directly from the cars into the smelting chamber, the hoppers serving merely as funnels through which the furnace draft created a downward suction, which drew most of the dust into the furnace, where it had a chance to

settle. The water-cooled plates protected the walls where the charge did not afford proper coverage.

The so-called "wet-charging" practice has not come into use in Montana, for the following reasons. In the first place the concentrates now being made, although almost three times as high in copper as formerly, still produce a very low grade of matte when charged directly to the smelting furnaces. It is cheaper to eliminate the excess sulfur in the roasters and the iron in the reverberatories than to perform these operations in the converters. Secondly, the charging of wet cold material into the smelting furnaces results in an increase in fuel requirements which may be estimated roundly at 50 per cent. Calculations as well as experience bear out the correctness of this figure. Third, the handling of wet sticky flotation concentrates is no easy task under the best of conditions and becomes almost impossible in the winter, when the thermometer often stands at 10° below zero.

These three penalties would be a heavy charge against the saving of some roasting expense and a little dusting. It is doubtful, moreover, whether the latter would be avoided, since arsenic, which is present in substantial percentage, would come off in the smelting furnace instead of in the roaster and must in either case be prevented from escaping into the air by treatment of the gases in Cottrells. Wet charging may be applied to good advantage where the concentrates are sufficiently high in copper to make a matte of acceptable grade and where they are free from contaminating volatile substances which must be kept out of the atmosphere. Obviously, where a new plant is under consideration, the possibility of simplifying the layout and saving the first cost of the roasters gives an added advantage to wet charging, which is lacking in an old plant that already has all such equipment and would probably have to spend some money to bypass it.

Controlling the Furnaces

At Anaconda, individual furnaces have at times smelted at the rate of 700 tons of solid charge per day, but no attempt has ever been made to push them to abnormal capacities, which it was contended could only be attained by shortening their life and thus adding to repair costs. Policy governing matters of that kind is often dictated by local conditions. At Anaconda, for example, there never has been a scarcity of furnaces; in fact, when selective flotation came into use, there was a superabundance of them. Hence the incentive was rather to carry on the most efficient all around operation giving due consideration to all of the factors that make up smelting costs, such as fuel ratio, copper content of slag, repairs to furnaces, etc.

There was, and is, also another controlling factor; that is, the need for limiting arsenical emanations from the main stack. At Anaconda,

there is one set of main Cottrell treaters through which pass the gases from all departments including the reverberatories. The draft is controlled with a view to securing the best possible clearances and sometimes this results in something of a deficiency at the reverberatories. It is never possible to run the furnaces in the free and uncontrolled manner that prevails in many other plants.

From early times, and particularly with the advent of the long furnaces, a great deal of attention was devoted to control of conditions governing combustion. In the grate-fired furnaces, the length of flame, its intensity and the composition of the exit gases were controlled by openings over the bridge wall, the size of which could be regulated. In the coal-dust fired furnaces, this regulation was accomplished by slides around the burners and by controlling the amount of air blown in with the coal or drawn in by the injector effect of the burner itself.

Analyses of the furnace gases were frequently made and at times continuous gas-analysis apparatus took samples and reported results every 15 minutes. This refinement was, however, scarcely worth the trouble of keeping the apparatus in repair, which was considerable. The CO_2 content of the outgoing gases was kept around 18 or 19 per cent and CO at less than 1 per cent. The importance of intensifying the flame in the smelting zone has always been recognized but it is important that the front or skimming end of the furnace be kept hot as well.

Smelting with Natural Gas

Early in September of 1931, natural gas from the fields in northern Montana was turned on at Anaconda and a few days later all smelting was being done with the new fuel. It was a success from the start and no serious difficulties were encountered. The furnaces were equipped with five so-called inspirator burners which are constructed and operate exactly like large Bunsen burners, air being drawn by the injector effect of the gas issuing from the nozzle, into the burner tube, through openings of which the size can be regulated. A regulator on each furnace reduces the 40-lb. line pressure to 20 lb. in the manifold preceding the burners. Separate 2-in. pipes lead from the gas manifold to each burner. Each burner has a separate control valve and pressure gage which makes it possible to operate all or part of the burners as conditions require. Each burner has a maximum capacity of 20,600 cu. ft. per hour, thus enabling 2,470,000 cu. ft. per 24 hr. to be delivered to the furnace if desired.

Natural gas has proved to be a very good fuel for reverberatory furnaces. It is clean, positive regulation is easily obtained, and the interior of the furnace is readily visible at all times. Thus charging is facilitated, as the operator can see where and when to charge without shutting off the gas. The wear on the roof is greatly lessened, thus

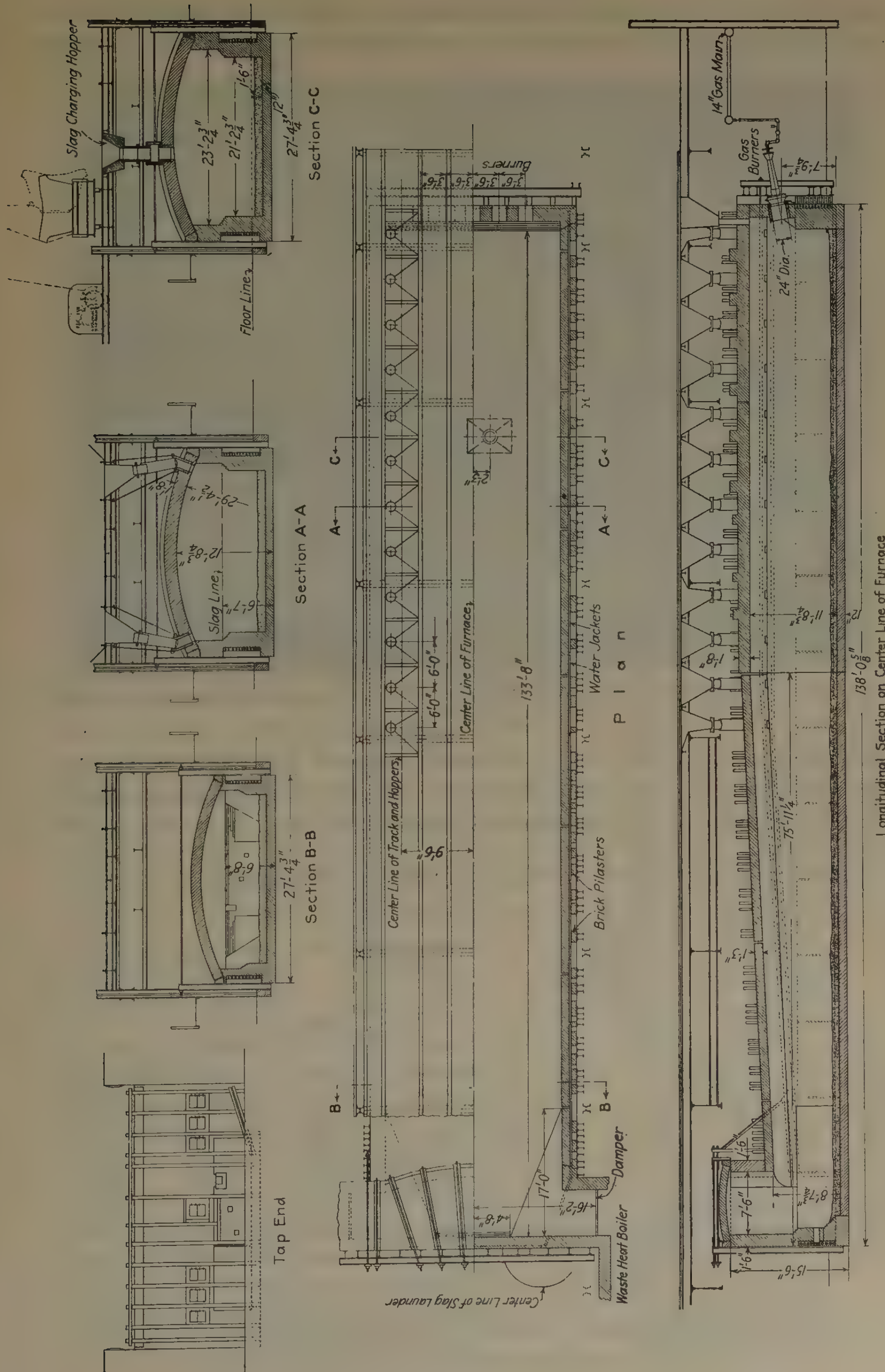

FIG. 43.—GAS-FIRED, WATER-COOLED ANACONDA REVERBERATORY SMELTING FURNACE OF 1931 TO DATE.

lengthening the campaigns and reducing repair costs. The performance of the furnaces with coal and gas is compared in Table 12. Periods

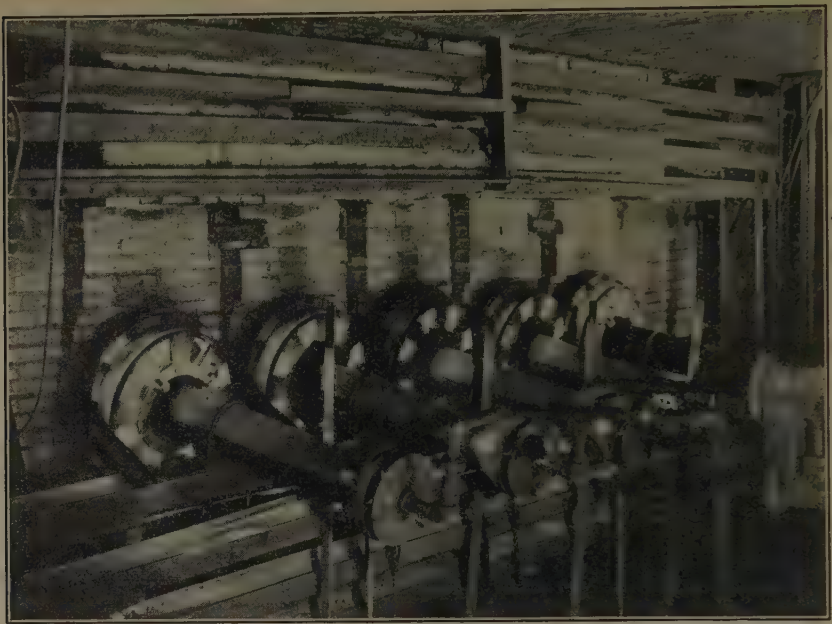

FIG. 44.—GAS BURNERS ON REVERBERATORY FURNACE AT ANACONDA REDUCTION WORKS.

compared are July 19 to Sept. 13, 1931, for coal and Sept. 28 to Nov. 22, 1931, for gas. An interval of two weeks is allowed for adjustment. During these two periods, smelting conditions were quite comparable

TABLE 12.—*Reverberatory-furnace Operations with Coal Dust and with Gas*

Week	Tons Smelted per Furnace Day		Tons Smelted per Ton, M Cu. Ft.		Hot Calcine in Charge, Per Cent	
	Coal	Gas	Coal	Gas	Coal	Gas
1st	551	541	6.51	0.265	50.5	46.8
2d	565	579	6.35	0.291	54.7	50.8
3d	576	592	6.48	0.303	53.1	55.2
4th	582	631	6.28	0.297	48.5	55.0
5th	602	665	6.38	0.295	51.8	53.7
6th	610	647	6.31	0.285	54.6	50.2
7th	597	643	6.16	0.284	50.5	49.7
8th	609	605	5.32[a]	0.258	48.2	47.3
Average	587	613	6.35	0.284	51.5	51.1

[a] Omitted from average on account of abnormal conditions.

and the furnaces treated molten converter slag to the extent of about 20 per cent of the solid charge. This is *not* included in the tons of material smelted per furnace day nor in tons smelted per million B.t.u.

During August and September the average per pound of wet coal was 11,750 B.t.u.; the average per cubic foot of gas during October and November was 1043 B.t.u. Hence there was required per ton of solid charge smelted:

For 8 weeks, July 19 to Sept. 13, 1931, using coal, 3,700,000 B.t.u. gross.

For 8 weeks, Sept. 28 to Nov. 22, 1931, using gas, 3,680,000 B.t.u. gross.

This comparison indicates that, all other conditions being equal, the quantity of material smelted per gross B.t.u. is the same for coal and gas. These eight weeks before and after changing to gas were selected because conditions were pretty much the same for both periods. The concentrates assayed about 25 per cent copper and the sulfur content of the charge was regulated to make a matte containing about 45 per cent copper. The variations in tonnage smelted from week to week are due not so much to furnace irregularities as to occasional shortages of material to be smelted, as the results were the average for the three furnaces in operation.

A calculation by F. F. Frick, research engineer at Anaconda, shows that the net B.t.u. in the coal were 96.6 per cent of the gross. The net B.t.u. in the gas were 91.3 per cent of the gross. For purposes of comparison in determining the relative effectiveness of different fuels, "net" effectiveness of different fuels, "net" B.t.u. values should be used, because the latent heat of steam condensed in the calorimeter, which is included in the "gross" figures, is obviously not of use in the furnace. On the basis of "net" B.t.u.,

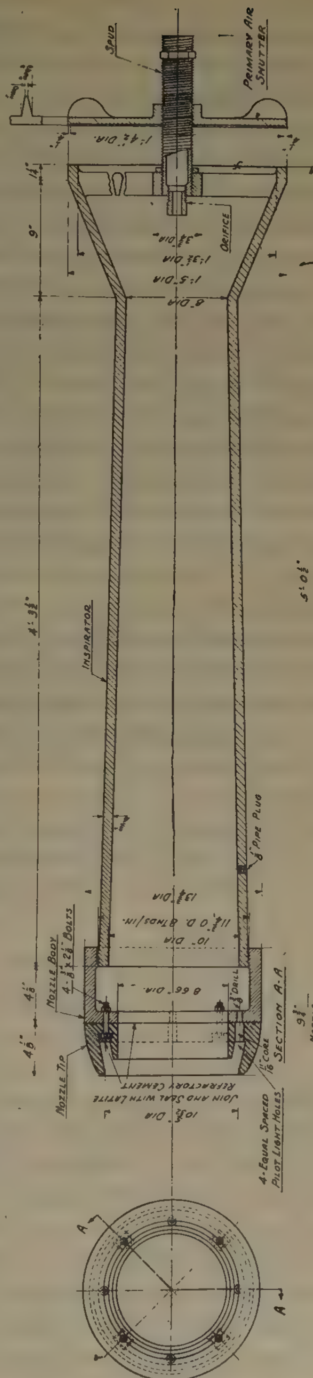

FIG. 45.—BURNER FOR NATURAL GAS, 1931 TO DATE.

the gas gave 5.8 per cent better results; that is, 5.8 per cent fewer heat units were required to smelt a ton of charge.

A comparison between the first six months of 1931, when the furnaces were operating on coal, with the first six months of 1932, when they ran on gas, gives 3,469,000 gross B.t.u. per ton of solid charge for coal, or 3,350,000 net; and 4,045,000 gross B.t.u. or 3,690,000 net, for gas. This represents 16.5 per cent more of gross or 10.7 per cent more of net B.t.u. for gas than for coal, but the difference is unquestionably due to variations in operating conditions which may be enumerated as follows:

During the first half of 1931, no molten converter slag was charged. This was treated in a special furnace at the converter plant, whereas during the first half of 1932 molten slag in the amount of 20 per cent of the solid charge was treated. Furthermore, during the first half of 1931, hot calcine made up 75.9 per cent of the total charge, whereas for the corresponding half of 1932 it amounted to only 48.6 per cent. "Wet slag mix," containing 4 to 5 per cent moisture, constituted 35.5 per cent of the charge in 1932, whereas cold converter slag and flotation concentrates, the chief ingredients of this mixture, made up only 17 per cent in 1931. The evaporation of moisture and heating the steam and cold charge to the temperature of the furnace absorb a large amount of heat. As has been previously stated, a reverberatory operating on cold wet charge requires approximately 50 per cent more fuel per ton of charge than the same furnace operating on dry hot charge.

A careful calculation by F. F. Frick reconciles the observed differences in B.t.u. required to smelt the ton of charge in the two periods as follows:

	Per Cent
Difference due to molten slag and hot calcine	8.00
Difference due to evaporation of water	1.90
Difference due to heating steam to furnace temperature	1.20
Total	11.10
Necessary to account for	10.70

It seems safe to conclude, therefore, that the smelting efficiency of gas is fully as great as that of coal when the comparison is on the basis of "net" B.t.u. and it is probable that it is greater. This conclusion is confirmed by results obtained at a plant in the Southwest when smelting a wet charge. Using oil, the gross heat units required were 5,376,000 B.t.u. per ton of charge as compared with 5,628,000 B.t.u. for gas, or 4.68 per cent more for gas. The theoretical difference between oil and gas due to water vapor is 4.28 per cent, which just about accounts for the observed difference in smelting efficiency between the two fuels. Other periods gave a balance of about 3 per cent in favor of gas, all of which bears out the results at Anaconda.

These observations seem to establish pretty definitely that there is not much in the idea formerly generally held, that flame luminosity

is important in reverberatory smelting. Gas burns with a nonluminous flame to such an extent that one can see from one end of the furnace to the other while the gas is on. Coal gives the maximum of luminosity, partly because of its high carbon content and partly because of the ash particles, which become incandescent. And yet it is apparent from the foregoing that such differences as have been observed and which at first glance might appear to favor the luminosity idea can be fully explained in other ways. When allowance is made for differences in the character of the material being smelted, it is evident that the amount of smelting that was done per heat unit with the nonluminous gas flame was at least as great as with the highly luminous coal flame.

Acknowledgments

In closing, I wish again to acknowledge my indebtedness to all who have contributed, through publications or directly, to the information contained in the foregoing pages, and I wish particularly to express my thanks to Mr. C. O. Scott, who made the excellent drawings that illustrate this article, and to Mr. R. B. Hanahan, who supervised their preparation.

I have endeavored to trace the progress of reverberatory matte smelting in Montana through the course of half a century, during which the capacity of the furnaces has been increased fiftyfold, the quantity of heat required to smelt a ton of charge has decreased to about one-third of what it was originally and the amount of necessary labor has been enormously reduced. These gratifying and rather marvelous achievements bear witness to the patience, skill and genius of the smeltermen of Montana. In part they are undoubtedly the result of the free and open interchange of ideas between metallurgists throughout the land, which has been so characteristic of American practice.

Reverberatory Smelting of Raw Concentrates at the International Smelter, Miami, Arizona

By P. D. I. Honeyman,* Miami, Ariz.

(New York Meeting, February, 1932)

Changing trends in concentration, involving regrinding with the subsequent production of a finely divided, high-grade flotation product, presents a real problem to the modern copper smelter. In the treatment of such a material, handling difficulties are numerous and potential possible losses may be very great. In many plants, of course, preliminary roasting of the furnace charge remains an economical necessity. Under such circumstances, the operator has no alternative except to follow orthodox roaster reverberatory practice, though taking all possible precautions to avoid excessive dust losses and bad working conditions. In other plants, roasting is not only unnecessary but would be highly undesirable, and at best the charge would be merely dried before entering the reverberatory furnace. In the latter case the possibilities of raw charging loom large.

The Miami plant of the International Smelting Co. handles the entire output of concentrates of the several mining companies operating in the Globe-Miami district. All of these companies produce a flotation concentrate which, at the Miami Copper Co. particularly, is all reground and is high in grade. In addition, the Inspiration Copper Co. ships a substantial tonnage of high-grade cement copper to the smelter from its leaching plant.

Following the flow sheet of the original plant, all such material received was bedded, dried in Wedge furnaces and smelted in reverberatory furnaces. At times during the history of the plant it has been necessary to carry the drying operation further, to a partial roast, before reverberatory smelting. Since the charge contained over 80 per cent flotation concentrates, the handling of which in a dried condition was at best a dusty job, unusual precautions were taken to cut down dust losses.

The Wedge furnace plant was equipped with a Cottrell treater and all hoppers and calcine cars were carefully designed to cut down dust loss in transferring the so-called calcine from hoppers to car and from car to furnace. In the furnace itself the dusting problem was a severe one. Arch erosion was very rapid and flues and waste-heat boilers rapidly filled up with a partly smelted flue dust.

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* Superintendent, Miami Plant, International Smelting Co.

Wet Charge Introduced at Miami

In 1927, following the success of feeding wet charge, as worked out by A. D. Wilkinson at Cananea, it was decided to try out wet charging at the International plant at Miami. Results were promising from the beginning, and the practice has been carried on and developed.

At present the charge is made up of approximately 77 per cent of concentrates, 4 per cent cement copper, 7.5 per cent flux and 11.5 per cent plant secondaries. An average analysis of the charge is as follows: Cu, 33.27 per cent; SiO_2, 9.95; Al_2O_3, 2.70; Fe, 21.68; CaO, 0.53; S, 24.82. This charge has an average moisture content of about 11 per cent and on occasions is decidedly sticky to handle.

It was, of course, evident that such a charge could not be handled in the usual calcine cars. The plant was fortunate in having available a surplus of regular concentrate cars which were impressed into the charge-car service. These cars are of the bin type, having a removable bottom made up of 2 by 8-in. boards, and hold 60 tons. The former calcine cars are used as locomotives to handle them to the feed floor and the combination works out very well. Enough cars are available so that all concentrate unloading may be done on the day shift, leaving sufficient stored charge in cars to carry the furnace plant through the ensuing 16 hr. One man only is employed in switching and unloading charge cars to the furnace bins and it is possible for him to handle more than 200 wet tons per shift.

Recently operations have been still further simplified by cutting out the bedding system, since it has been found practical to bed directly in the charge cars themselves. Concentrate cars to be unloaded are spotted as desired in the unloading pocket and unloaded on to a conveyor belt. Flux is fed to the same belt, the common stream discharging to the charge car, which is moved slowly back and forth for its entire length until it is filled up. The bedding bins merely act as surge bins to take up an unusual influx of tonnage over and above the operating rate of the plant.

Furnace Plant

The reverberatory furnace plant consists of four furnaces of which at present only one is being operated. Three of these furnaces are 21 ft. wide, while the fourth, a larger furnace, is 25 ft. wide. All furnaces are 120 ft. long. Hot gases discharge from the uptakes to a header flue and thence to the waste-heat boilers.

As designed for calcine smelting, the furnaces were equipped with drag chain conveyors which dragged the calcine from bins located under the feed floor and discharged through various gate-controlled feed pipes to the furnace. When wet charging was first thought of it was decided to

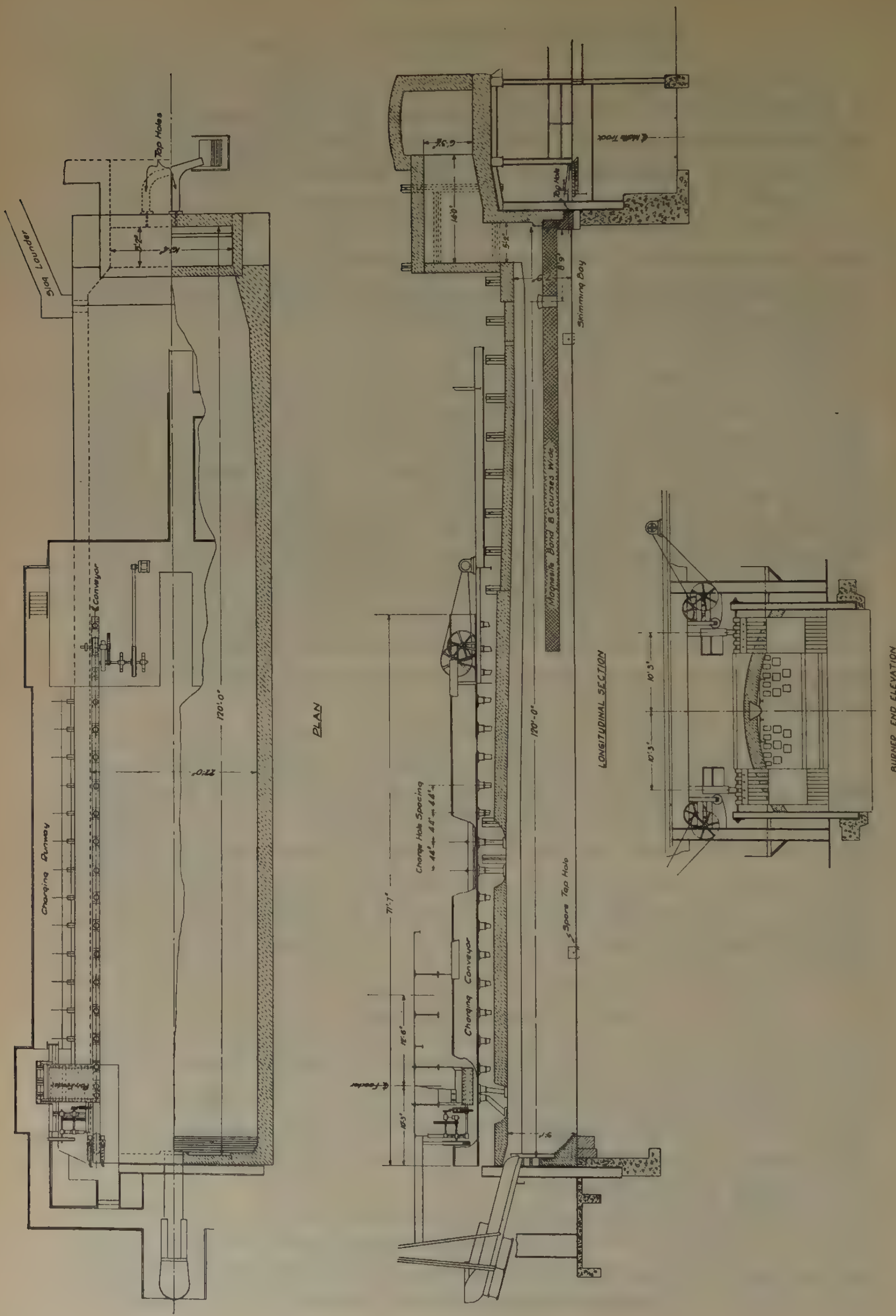

FIG. 1.—REVERBERATORY FURNACE NO. 1, INTERNATIONAL SMELTING CO., INSPIRATION, ARIZONA.

utilize this same system, with certain necessary modifications, for the handling of this wet feed. Results obtained were more or less satisfactory, though it was soon evident that many mechanical changes would have to be made to assure satisfactory feeding. After due consideration, it was decided that perhaps better results in feeding could be obtained by making use of a belt conveyor and tripper car. One furnace accordingly was equipped in this manner and was started up. While many of the handling difficulties associated with the chain conveyor were overcome, the furnace did not do so well and a higher fuel ratio was obtained than in the chain-fed furnace. A study of reasons led to the belief that the discharge of a large, steady stream of charge from the belt through an individual charge hole tended to pack the charge pile and cut off the hole before sufficient charge had worked its way on to the surface of the bath. In contrast, the drag chain fed a thin stream through several holes simultaneously, a "trickle feed" which flooded the bath and increased the size of the individual charges it was possible to put into the furnace at one time.

The final conclusion was that a properly designed drag chain conveyor system was the best way to feed wet charge to the furnace. This system was thus adhered to in making subsequent furnace changes. Every effort was made to capitalize on early experience and all early difficulties encountered were studied in making such changes.

Furnaces are now equipped with charge bins under the feed floor, which feed by way of a pan feeder to the drag chain conveyor. Regular controlled feed to the conveyor is essential to avoid overloading and choking down the drag chain. Charge pipes, 8-in. dia., are spaced at 44-in. intervals and are gate-controlled from the charging runway. All charge holes are equipped with peepholes through which the condition of the charge within the furnace may be observed. It is necessary, also, for the charger to assist the flow of charge into the furnace by means of a short rod worked through this peephole. In order to cut down the amount of "drag over," the conveyor is equipped with heavy scrapers which serve to keep the chain clean. Both chain and feeder motors are controlled by push buttons conveniently located along the charging runways.

Charging Routine

In charging, the charge car is spotted over the furnace bin and feeder and conveyor are both started up after all slides have been opened. Since the furnace is run on practically a balanced draft, there is not a great deal of cold air infiltration through these opened charge holes; in fact, generally there is a considerable back-pressure. Starting from the firing end, the operator feeds his charge into two or three holes at a time until such holes are filled up, when he passes along to the next series of

holes, eventually covering the entire charging zone, which extends about 65 ft. down the furnace.

Quite a detailed study was made to determine the most satisfactory charging routine. Many controversial questions were raised relative to the size and frequency of charges. Much experimental work was done and all methods, from continuous feeding of a small stream to very intermittent feeding of very large charges, were tried out. It was eventually decided that best results were obtained by feeding comparatively large charges about six times per shift. The frequency and size of charges will depend to some extent, however, upon the nature of the charge. A deep molten bath is maintained between the two charge piles in the smelting zone and every effort is made to get as much charge as possible down to the surface of the bath. Failure to achieve this result invariably slows up the furnace. Very little trouble has been experienced from explosions as the wet charge enters the bath. Some shooting has taken place, particularly if the bath did not have a sufficient depth of molten slag on top of the matte.

Frequently the furnace has been run at a rate of more than 600 wet tons per day, which means that charges often exceed 60,000 lb. in weight. Generally it takes from 20 to 30 min. to charge the furnace completely.

Much assistance was obtained in working out charging problems by using a recording wattmeter which, when hooked in the feeder motor circuit, presented an accurate recorded diagram of charging throughout the 24 hours.

As before mentioned, the average moisture content of the charge is about 11 to 12 per cent, though charges as high as 15 per cent moisture have been handled. As the moisture goes up, however, the charge rapidly becomes sticky and is much more difficult to handle.

Molten converter slag is returned to the furnace through a launder discharging through the middle of the bridge wall. The action of converter slag is very marked in the furnace at times and the reaction is sufficient to cause a violent boiling in the bath which may extend the whole length of the charging zone. This boiling action, of course, materially assists in the mixing of the surface charge with the molten bath and undoubtedly assists smelting. It is, too, the one visible sign of reaction which accounts to some extent for the much smaller accumulation of magnetite in a wet-charge furnace than in a calcine-fed furnace.

Fuel Consumption

Furnaces are oil-fired and, as would be expected, the gross fuel consumption per ton of dry solid charge is considerably higher than would be obtained in smelting a dried or roasted charge. This higher gross fuel consumption in the reverberatory furnace is offset, however, by an increased waste-heat recovery, together with the oil saved by dispensing

with the drier operation. A comparison of net fuel consumption, over three separate periods of distinctive operation, is interesting (Table 1).

Necessarily, of course, with this high fuel ratio, it follows that in order to maintain the daily furnace tonnage considerably more oil must be burned. At a normal operating rate of 500 dry tons per furnace day, the fuel consumption is around 450 bbl. This may increase or decrease as the daily tonnage rate fluctuates. Being so much higher than the consumption rate for calcine smelting, more than a few problems of combustion control were encountered before satisfaction was obtained.

TABLE 1.—*Fuel Consumption with Three Methods of Charging*

	1931, 6 Mos. Wet Charge, Bbl.	1924, Yr. Dry Charge, Bbl.	1926, Yr. Partial Roast, Bbl.
Gross fuel ratio per ton dry solid charge at reverberatory furnaces	0.902	0.667	0.577
Equivalent waste-heat steam recovery	0.470	0.263	0.219
Net fuel ratio per ton dry solid charge at reverberatory furnaces	0.432	0.394	0.358
Drier fuel consumption		0.105	0.077
Net total plant fuel ratio per dry ton smelted	0.432	0.499	0.435

Oil is burned in a battery of Cananea-type low-pressure burners. Generally, 10 burners are utilized, though this number may be increased to 12 or reduced to 8, according to the rate of fuel consumption. Primary air is supplied at 40-oz. pressure to the burners and the secondary air supply is closely controlled by adjustment of auxiliary air ports, with which the bridge wall is well equipped.

A short flame and good mixing are essential to insure complete combustion of fuel within the furnace. Proper control is maintained by daily gas analyses and the usual sample taken about 10 ft. back of the verb arch shows a trace of CO in the gases.

The furnace is run on a very low draft, averaging about 0.02 in. of water, the reading being taken through the verb arch. Low draft is desirable to reduce to a minimum infiltration of cold air through furnace walls, flues and waste-heat boiler settings, and makes for steadier control. While the importance of this fact was early recognized, much difficulty was encountered in establishing proper conditions until the size of the furnace uptake was materially increased. By increasing its width to 16 ft. and thus enlarging the cross-sectional area to 82 sq. ft., much better conditions were established. A steady draft is maintained during both charging and smelting periods by proper adjustment of boiler dampers for both periods.

Waste-heat Gases

Waste-heat gases discharge from the uptake to a header flue common to four waste-heat boilers, all of which are kept in service. Steam is generated at 220 lb. pressure and superheated approximately 50°. Boiler exit gases have a temperature around 550° F. Every effort is made to keep boiler settings tight and the infiltration of cold air is held down to a minimum.

As before mentioned, the waste-heat recovery, based on the equivalent heat value of the steam, is much higher than in smelting dried or roasted charge, and amounts to 52.1 per cent of the fuel burned, in contrast to 39.1 per cent for dried charge and 37.9 per cent for partly roasted charge. Undoubtedly a great deal of heat is lost in heating up header flues and boiler cross-over flues, which would be saved by a more up-to-date arrangement of boilers in relation to the furnace. The actual unit evaporation is 6.9 and the equivalent unit evaporation is 7.75 lb. of water per pound of oil burned. Very much less dust accumulates in the waste-heat boilers when wet-charging, and the problem of keeping boilers clean is a comparatively simple one.

The elimination of sulfur from the furnace charges varies from 25 to 30 per cent; the second figure closely approximates the elimination obtained when smelting a dried charge. While both physical and chemical characteristics of the charge play a part in determining the elimination of sulfur, a study of the mineral constituents of the charge brings out the fact that normally the sulfur driven off is about 85 per cent of the free-atom sulfur contained in the charge.

Tapping

In deciding on the location of tap-holes in the furnace for the removal of matte, consideration was given to the possible dangers of tapping through the side wall under a pile of wet charge in the smelting zone. In order to avoid any potential danger, as well as to assist in keeping the front end of the furnace clear of magnetite accumulation, it was decided to install the tap-holes well toward the front end of the furnace, and they were put in just back of the verb arch. Later on further changes were made and today matte is being tapped directly through the front wall of the furnace to ladles held on a transfer car spotted on the inner track of the slag tunnel. This, of course, necessitates tramming of the hot matte to the converter aisle, a proceeding which at first glance might appear awkward and complicated. As a matter of fact, the arrangement has proved to be highly satisfactory and, although it has since been found quite practical to tap through the charge pile, in the smelting zone, no change has ever been contemplated since the front wall holes were put in. The matte, beyond question of doubt, taps more freely here than at any

other tap-hole location in the furnace, in spite of the fact that in recent months higher and still higher grade of charge has resulted in the production of a matte containing as much as 55 per cent copper.

Also, there is no doubt that the maintenance of circulation in the body of matte toward the front of the furnace plays a large part in helping to keep the bottom free of accumulations which in the past have been responsible for the early termination of a furnace campaign. The campaign of our present operating furnace has been extended over two years, with no apparent building up of the bottom at the front end. Regardless of tap-hole location, however, a great reduction in the amount of magnetite accumulations is one of the features markedly noticeable in a furnace operated on wet charge.

One reason that led to the adoption of the front-wall tapping arrangement was the desire to relocate the skimming bay in the side wall. The thought was to get away from fouling the slag by drippings from the uptake which, while very much minimized by the reduction of dusting, still proved to be a source of trouble. Thus the present furnaces are skimmed through the side wall just back of the verb arch. Results achieved have been eminently satisfactory and no handling troubles have developed even though the slag launder is 39 ft. long.

Analyses of matte and slag produced during a recent six months' period are as follows:

	Cu, Per Cent	SiO_2, Per Cent	Al_2O_3, Per Cent	Fe, Per Cent	FeO, Per Cent	CaO, Per Cent	S, Per Cent
Matte.........	45.9			26.0			25.2
Dump slag....	0.52	34.5	7.7		49.8	1.26	

Slag loss ratio, 1.13.

Furnace Brick and Repairs

The furnaces are of the usual type, with practically straight-line arches, constructed largely of silica brick. Built into the walls, however, from the end of the charge line to the front of the furnace, is a six-course band of magnesite brick. The front wall, from the foundation to above the normal slag line, is also faced with magnesite, as is the bridge wall along the slag line. In building the furnace, charge holes were located about 1 ft. in from the side walls, and since the furnace is always kept full, the upper side wall is never exposed to erosion. Consequently, in more than two years of continuous operation, no repairs of any sort have been made to the walls. Furthermore, in making arch replacements it has not been necessary to replace more than to within three courses of the skewback bricks. The arch is of 20-in. brick throughout, with the

exception of one 20-ft. section in the smelting zone, which has a 15-in. center, 10 ft. wide. Since the heaviest erosion is toward the line of charge holes, the stepped off arch results in more uniform wear of the entire width of arch.

Furnace repairs have been much reduced since the adoption of wet charging. Decreased dusting within the furnace is undoubtedly responsible for this betterment. During the period of calcine smelting, when dusting conditions were particularly difficult, the wear and tear on furnace brickwork was very severe. Under such conditions a campaign rarely extended over nine months, at the end of which time it was necessary to shut down the furnace for a general repair. Such a repair entailed a considerable wall and arch replacement, together with a more or less extensive rebuilding of the header flue. In addition, during the campaign, at least two arch replacements were necessary in the smelting zone. Such "hot patch" replacements involved the rebuilding of from 20 to 40 ft. of arch and upper side walls. It was generally considered that for calcine a 90-day run for that section of arch, in the smelting zone, was a good performance.

The present campaign of our operating furnace has extended over two years since the last general repair, during which time the furnace has never completely lost its heat. During this time, of course, several hot-patch arch replacements have been necessary in the smelting zone and some repairs of major importance were made to the flue system. It is noticeable, however, that the length of run of smelting-zone arch has been extended from 90 days to from six to nine months. Furthermore, the arch in the front half of the furnace shows few signs of wear and probably is still good for a long time to come. As has been mentioned, no repairs have been made to side walls during the campaign.

Advantages of Wet Smelting

Reverberatory costs with wet charge compare favorably with costs achieved in former times when smelting a dried charge, in spite of the fact that operations have been reduced to a partial one-furnace basis.

In general, it may be said that where plant conditions warrant the adoption of wet smelting many benefits are to be gained therefrom. The practical elimination of the dusting problem greatly simplifies the entire smelting operation. Working conditions are improved; losses are reduced and the wear and tear on furnace brickwork, resulting in periodic repair shutdowns and short campaigns, are very much lessened.

The utilization of waste heat and the maintenance of an economical grade of matte are special individual plant problems to be studied by the individual operator and given due prominence in considering the applicability of wet charging to his particular smelting problem.

DISCUSSION

(Francis R. Pyne presiding)

C. R. Hayward, Cambridge, Mass.—Is there any definite information as to how low in grade the concentrates could be and still make it advisable to smelt in this way? Your concentrates are a little higher than the average, which makes the process rather favorable under your conditions. One or two plants are using cold concentrates and it is a matter of some interest as to how general this practice is likely to become. Probably it is really dependent on the copper and sulfur content of the concentrates, but what are the economical limits?

M. S. Mazany, Montvale, N. J.—In carrying on an operation of this kind, it would be important to know the mechanical condition of the concentrates; whether they are from acid or alkaline flotation circuits, particularly in view of the aluminous content of the concentrates, inasmuch as with increased aluminous content there is a tendency for the oil from an acid circuit to form a putty with the clay content of the concentrates, which would make the material very sticky. The charging, as a result, would be exceedingly difficult.

H. W. Mossman, McGill, Nev. (written discussion).—It should be remembered that the belt conveyor and tripper car system that Mr. Honeyman mentions was primarily a gravity feed of poor design, and to only a minor extent really a belt conveyor and tripper system. I saw a testing engineer, B. J. Skoblin, make an investigation for Mr. Honeyman on the weights of feed handled by gravity and by belt; and in May, 1929, I protested to Mr. Honeyman about his use of the hybrid arrangement. Then, too, he makes no mention of another belt conveyor which, under normal operating conditions, made a better gross fuel ratio in a 21-ft. furnace than a drag chain conveyor did on the last previous campaign of the 25-ft. furnace, for similar parts of the two runs.

The two systems that Mr. Honeyman compares are those put in on the No. 3 and No. 2 reverberatories in 1927. Two furnaces were equipped with the hybrid gravity-belt system. The drag chain was used in the summer of 1927, and the gravity-belt system in the latter part of 1927 and the first part of 1928. The charge floor is very low, and had insufficient headroom for a large tripper car back towards the burners where most of the charge is smelted, so the gravity feed, into six chutes on each side of the furnace, had to be used to take care of that part of the furnace. These chutes served the first 28 ft. in front of the burners. As I remember the notes made by Mr. Skoblin, the gravity feed was about 65 per cent of the total. I have an accurate record of the feed put in at each charge hole for the next campaign of the No. 3 furnace, and then the first 28 ft. smelted 60.6 per cent of the total. Raw concentrates were dropped into these chutes from standard-size concentrate cars, the feed dropping an average of 22 ft. into the furnace. The concentrates, falling so far, and under very poor control, rolled into the channel of the furnace to some extent. The partly plugged furnace, with a few large, irregular piles near the burners, smelted slowly. The conveyor belt, discharging by a tripper, was used out at the tail end of the charge zone only, where 39.4 per cent of the charge was smelted. Such a gravity-and-belt system is not typical of 100 per cent belt conveyors.

Another belt charging system, which the author does not mention, was used on the No. 3 reverberatory in 1929. A reversible shuttle belt conveyor was employed. This conveyor was low enough to go under the charge floor, and delivered feed to the entire charge zone. Miami is an Anaconda subsidiary, and Anaconda has secured patent No. 1857592 on this belt system, with myself as one of the inventors. The system has very advanced design in relation to automatic control of the distribution of the feed.

The conveyor moved rapidly from one charge hole to another, putting in exactly the right amount at each place, and bedded down the charge in several thin strata. The amount put in at each hole was controlled by a mechanism, which is in effect an electromechanical bar graph of the smelting rate at each charge hole.

It is difficult to get comparisons of furnace operation at Miami. The 21-ft. furnaces are not as good as the 25-ft. furnace, so acceptable comparisons must take into account which furnaces are operating. Also, the relative parts of the furnace campaigns used for comparisons must be taken into account, as well as the absolute values of the figures used. There is a fairly steady improvement in the gross fuel ratio in a normally operated furnace as the campaign goes on. For instance, in the 1928–29 campaign of the 25-ft. furnace on raw concentrates, the gross fuel ratio for the first month was 0.99 bbl. of oil per ton of charge. The gross fuel ratio for the eighth month was 0.82.

I obtained from the original records the figures for the operation of the automatic belt system during its first three months. I have been told that at the end of that period a charge of flotation cell cleanup almost froze up the furnace, and the immense amount of fuel used to thaw out the furnace spoils the data from there on for comparative purposes. The first three months of the run on the 21-ft. furnace with the automatic belt system compared with the first three months' run of the 25-ft. furnace with a drag chain conveyor shows a gross fuel ratio of 0.957 for the belt and 0.963 for the drag chain. When a 100 per cent belt conveyor, with the automatic control of distribution, made a small furnace do better than a large furnace with the drag chain, it is reasonable to infer that on furnaces of the same size, on normal operating conditions, the belt conveyor would make even a better showing.

P. D. I. Honeyman (written discussion).—It is rather difficult to give a definite answer to Mr. Hayward's question. It seems to me that the economical limits of the method must be determined by the individual plant. As he says, the copper and sulfur content of the charge are to a large extent the controlling factors and where such factors result in the production of a low-grade matte in "raw" smelting converting costs would be increased. The balancing of higher converting costs against the eliminated roaster cost, decreased furnace-repair costs, and savings made as a result of decreased losses would tell the story. As mentioned in the paper, efficient and economical disposal of waste heat is another important point governing the economics of wet smelting.

Mr. Mazany's remarks are pertinent, and there is no doubt that in considering wet smelting thought must be given to the physical nature of the concentrates. Very sticky concentrates would complicate charging problems and might require special handling. However, the trend of modern concentration is toward low insoluble concentrates, which are not difficult to handle, and in any case the addition of normal amounts of flux and plant secondaries to the charge would tend to make such a sticky concentrate somewhat easier to handle.

Development of Gun-feed Reverberatory Furnaces at Garfield Plant of American Smelting and Refining Co.*

By R. A. Wagstaff, Garfield, Utah†

(New York Meeting, February, 1932)

The method of charging a reverberatory furnace has changed many times since smelting was introduced in this country from the old smelters of Swansea, England. The cause of the latest change at the Garfield plant has been the excessive fineness of the charge due to fine grinding in the concentration of the crude copper ores.

The many desirable features of deep-bath smelting as now used at Garfield have been proved. The old method of dropping the charge through holes in the arch (known as center-feed method) causes excessive dusting which makes heavy slag and flue-dust losses, also rapid deterioration of the silica brickwork of the furnace through fusion with the iron oxides from the dust.

The purpose of this paper is to describe a gun feeder that was designed to overcome the disadvantages of the older methods of center feeding and still maintain the good features of deep-bath smelting with the hot finer charge.

A gun feeder was designed to introduce the finely divided hot calcines under the moving gas stream of the combustion zone and still have them spread in a uniform blanket over the highly heated matte bath. The temperature of the furnace was so great that it was impossible to maintain a stationary feeder, so a movable feeder was designed that could be introduced into the highly heated zone and be withdrawn after the charge had been dropped. It was necessary to protect the nozzle of the gun with a water-cooled coil. Many different types of protected nozzles were tried, such as air jackets, water jackets and heat-resisting steel alloys, but none proved satisfactory.

The angle of inclination of the feeder was found to be an important factor in spreading the charge evenly over the surface of the bath. An angle of 34° to 37° was found to be the most advantageous. To allow a continuous flow of ore from the charge car to the point of discharge inside the furnace, the feeder was made with telescopic connections. The pipe leading from the car is stationary and forms the inside part of the telescope. The outside section of the telescope is movable, and is

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* Assistant Superintendent, Utah Dept., American Smelting & Refining Co.

known as the gun, or feeder. This part is made up of two sections; the upper section is the carriage, the lower section is the water-cooled nose. The lower section is removable and can be readily changed should anything happen to the water cooling system. The upper section carries the driving mechanism, which consists of a rack and pinion. The pinion is driven by a small motor, jaw clutches being used to transmit the power. The gun carriage rides on tracks supported from above. A double track keeps the gun from tipping and the flanged wheels of the carriage prevent side play. Counter-weights are used to allow easy movement of the gun in and out of the furnace. All bearings are equipped with Timken bearings and Alemite greasing system.

Five guns are used on a furnace, two on one side and three on the other. The ports are staggered on opposite sides to allow even distribution of the charge. Past experience has shown that if drop holes are eliminated from the arch of the furnace it will last much longer. To gain this advantage the gun feeder was made to enter the furnace through the side wall, which also permitted the installation of a simple counterweighted water-cooled gate, which keeps out excess cold air and reduces radiation losses.

The simplicity of design and the sturdy construction of the gun have kept the operating and mechanical cost down to a minimum. The regular furnace crew is able to operate the gun feeder with little or no training. It is beneficial to shift the disposition and rotation of the charges in the smelting zone of the furnace to satisfy the nature of the ore to be smelted. This can be easily learned by the operators. One of the important features of this method of feeding is that the operator must know the condition of his furnace at all times, there can be no guesswork as to where and when charging is to be done.

The first experiments on this method of feeding were tried in 1928 and 1929. One furnace was equipped with a crude "layout" that gave promising results. Lower flue-dust losses were obtained and the life of the brickwork of the furnace was much longer. This preliminary work was followed by more complete installations on two other furnaces. At present the Garfield plant has three furnaces completely equipped with this method of feeding, the results of which have been very gratifying.

Comparison of Dust Losses

During the development of the gun feed some interesting data were collected (Table 1) showing comparative dust losses on reverberatory furnaces using different methods of feeding. Two interesting factors are brought out in these comparisons; i.e.,

1. The amount of dust produced with a gun-feed furnace is less than with the other types of furnaces.

2. The copper content of the dust produced on a gun-fed furnace is lower in copper content than with the other types of furnaces, when the copper content of the original charge is taken into consideration.

TABLE 1.—*Comparison of Feeds for Reverberatory Furnaces*

Type of Furnace	Solid Tons Smelted per Day	Dust Leaving Furnace per Day, Tons	Dust Produced, Per Cent of Tons Smelted	Copper Content of Charge, Per Cent	Copper Content of Dust, Per Cent	Duration of Test, Days	Size of Furnace, Sq. Ft.
SIDE AND CENTER FEED							
Side feed	835	9.12	1.091	14.4	8.1	29	2733
Center feed	838	11.39	1.359	14.4	8.9	29	2728
GUN AND CENTER FEED							
Gun feed	847	8.29	0.98	26.6	14.5	29	2733
Center feed	809	12.29	1.52	26.6	19.5	29	2728

COMPARISON OF FURNACE LIFE

One of the chief factors that encouraged the design of a gun feed was the hope of extending the average campaign of a reverberatory furnace. Owing to factors already mentioned, the life of a center-feed furnace smelting an average of 800 tons per day was 80 to 100 days. By clever patching this could be extended another 10 to 20 days, but that was the limit. The last campaign on a gun-feed furnace has lasted more than 200 days and smelted more than 200,000 tons of charge.

FURNACE CONSTRUCTION

While progress was being made in perfecting this type of feed, certain weaknesses in previous furnace design were appreciated and corrected. Among them was the side wall. It was found desirable to build a much heavier foundation of magnesite brick. On this heavy foundation a silica wall of panel construction was developed that could be cheaply patched and replaced. The upper section or skewback portion of the side wall was strengthened, water coils were placed in the skewback plates to protect the buckstays from the heat in the protracted campaigns (Fig. 1).

Ribbed-arch construction had been well developed during center-feed furnace operation and this fitted in nicely on the gun-feed furnace. The open construction around the arch of a gun-feed furnace made it possible to get away from the extraordinary hot repair conditions that are common to both center and side-feed reverberatory furnaces. The

gun feed allows an accessible arch, easy to repair and free from calcine accumulations.

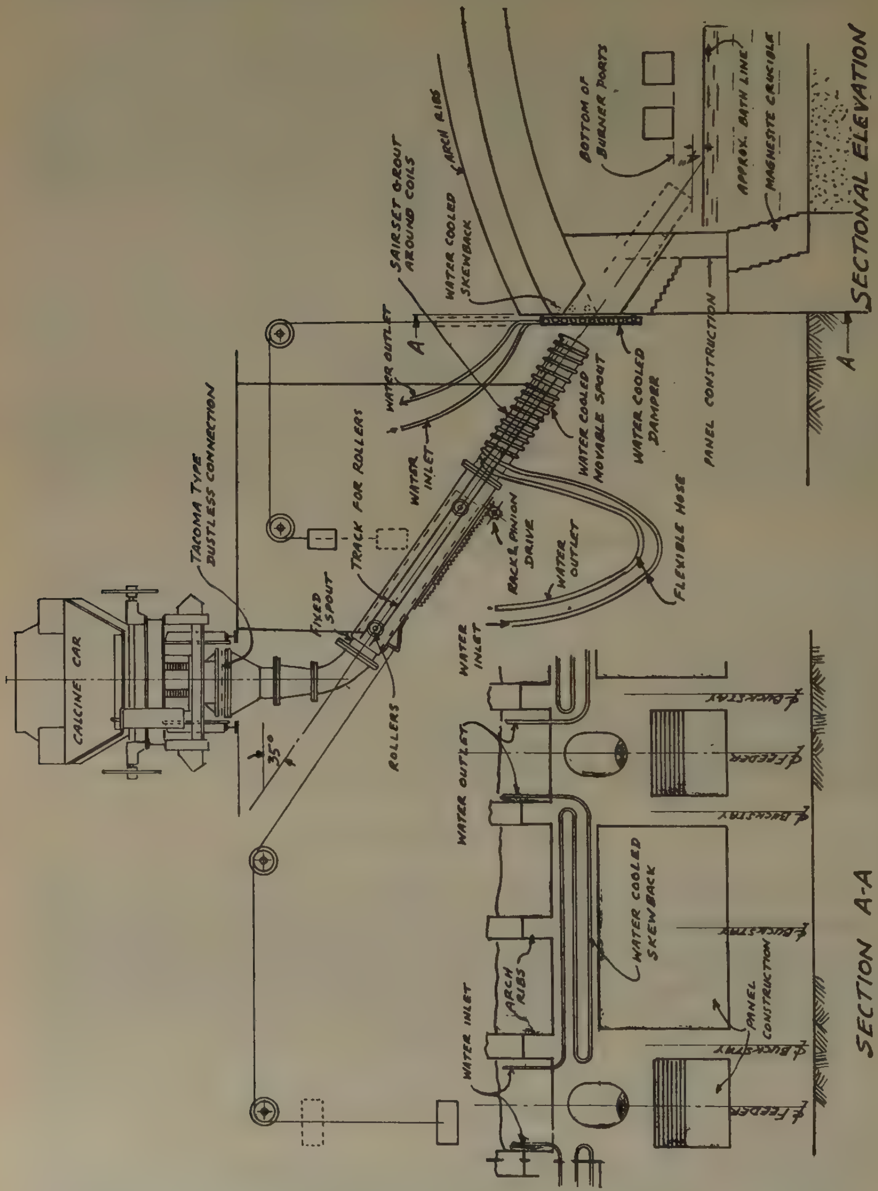

FIG. 1.—PRINCIPAL FEATURES OF GUN FEEDER AND CONSTRUCTION OF SIDE WALL.

The question of why a deep-bath reverberatory furnace is the better suited for a "hot finely divided charge" will surely arise. This question can only be answered by years of experience in reverberatory smelting. The main reasons for a preference are:

1. Lower metal losses.
2. Better fuel ratios.
3. Easier manipulation of furnace operations, *viz:*
   - A. Control of matte supply, with large available bath.
   - B. Control of copper fluctuations in matte, with large matte bath.
   - C. Simplicity of getting proper mixture of converter slag and smelting charge, so that a homogeneous slag may result.
   - D. Hot "wild" calcines are not suitable for proper side-feed charging, or side-wall protection.
   - E. Easier combustion regulation, due to uniform smelting zone area.
   - F. Better furnace feeding regulation, because furnace conditions can be sized up more quickly both by observation and physical inspection.
   - G. Relief from high furnace bottoms; an ease of keeping furnace in good smelting condition at all times.

The only factor that can offset these favorable conditions in deep-bath smelting is an increase in repair costs and the gun feed and the improvements in arch and side-wall construction have decreased the repair cost to such an extent that it is now possible to have all the good features of a deep-bath smelting furnace without this objection.

DISCUSSION

(*Francis R. Pyne presiding*)

C. R. Kuzell, Clarkdale, Ariz. (written discussion).—We copper metallurgists are indebted to Mr. Wagstaff for the presentation of this paper at this time because it pertains to one of the two successful methods of coping with the problem introduced by the fact that in recent years the feed for reverberatory furnaces has become much higher in grade and much finer in size. This was a natural result of the growth in the use of the flotation process and in the progress of finer grinding of ores. The engineer confronted with the building of a new smelter today would have to give serious consideration to the choice between wet smelting of the raw concentrate and such devices as Mr. Wagstaff's for the introduction of the hot and wild roasted concentrates. Both are being done successfully.

Mr. Wagstaff's scheme apparently goes a long way toward the elimination of dusting due to the introduction of the hot calcine and has the further advantage of good distribution of the charge over the bath. It is obvious that this method makes it much easier to maintain a deep bath, so that operators who prefer such a smelting method will have much less difficulty than with side charging. Also, the introduction of an admixture of molten converter slag with the bath of the reverberatory furnace can be much more easily accomplished. It would appear that the net cross-sectional area available for combustion space is increased and that therefore more capacity might be obtained per unit of floor area.

Very likely engineers may find that wet smelting will fit certain conditions as well as or better than this improved method of smelting calcined concentrates, and the reverse may also be true for other conditions; it would be interesting to know whether Mr. Wagstaff can present comparative data on the two methods as applied to the same kind of concentrates.

An Experimental Combination of Shaft Roasting and Reverberatory Smelting

By Frederick Laist,* New York, N. Y., and J. P. Cooper,† Anaconda, Mont.

(New York Meeting, February, 1934)

In the spring of 1931 an experiment was made at Anaconda with a small reverberatory furnace, with which was combined a roasting shaft. The arrangement is shown in the accompanying drawing (Fig. 1) and requires but little explanation. The general idea was to roast flotation concentrates, mixed with the proper fluxing materials, by dropping them down a heated shaft, from which the calcines would enter the smelting zone of the reverberatory furnace through an opening in the roof.

The hearth of the reverberatory was 3 ft. wide by 21 ft. long. The roasting shaft was 3 ft. square inside and had an effective height of about 20 ft. The reverberatory furnace was arranged so that it could be charged either directly through the roof with regular hot calcines, in the usual manner, or by means of the shaft, at the top of which were a feeding arrangement by which cold flotation concentrates, suitably fluxed, could be fed continuously and a pipe through which cold air was blown in. Means were provided for distributing the feed over the cross-section of the roasting shaft. At first a mechanical arrangement was used but later it was found that a simple cone served the purpose equally well and gave less operating trouble.

On account of the small size of the equipment, it was realized that absolute capacity and fuel consumption figures would mean but little. It was not to be expected that ratios comparable to those obtained in large furnaces could be gotten, and the only way in which any idea could be obtained as to the possibilities of the combination furnace was to run it in parallel with the reverberatory furnace charged in the usual manner and operating without the shaft. The results of such parallel runs are given in Table 1. The amount of oil burned in the reverberatory was purposely maintained at substantially the same rate during both runs in order that the benefits resulting from the shaft might be gaged.

It was expected that the shaft would considerably increase the capacity of the reverberatory and that a better fuel ratio would be obtained on account of the high temperature at which the calcines would enter the

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smelting region. Inasmuch as the shaft was down-drafted and the hot gases were drawn through the reverberatory chamber, it was thought that some additional benefit might result from the heat contained in these gases. It was feared that there might be considerable trouble from the building up of accretions in the roasting shaft, particularly at the point of entry into the reverberatory furnace. However, no troubles of this kind were encountered so long as the temperature of the calcines at the throat of the shaft was kept below 1500° F. On account of the small size of the shaft, there was some uncertainty as to whether it would maintain its heat. As a matter of fact, no oil whatever was required for roasting and even after a shutdown of from 30 to 60 minutes, the shaft heat could be brought back without the use of oil.

The shaft was operated both in combination with the reverberatory furnace and alone. When connected with the furnace, its capacity was necessarily limited by the rate at which smelting proceeded. It was soon apparent that the shaft had a much greater capacity and it was desirable to determine what this might be. Therefore arrangements were made for bypassing the reverberatory, so that the shaft could be operated unhampered, and at the same time the effect of moisture in the feed was determined. Analyses were made of the various products, all of which data are shown in a summarized way in the tabulations herewith.

It was expected that the dust losses from the reverberatory would increase when operated in conjunction with the shaft but it was thought that losses of this kind would not be excessive and would probably be much less in a large furnace than in the small one. As a matter of fact, the copper in the dust leaving the reverberatory furnace averaged less than 2.5 per cent of the total input. Considering the short distance between the flue and the point where the hot calcine entered the furnace, and also considering the fact that the calcine dropped through the flame, this result was satisfactory. Because of the high temperature of the flue gases (1950° F.), and for other reasons, the dust determinations were difficult but probably indicate fairly well the true state of affairs. The furnace temperature was about 2300° F. and from the time the furnace was started until it was shut down about 94 per cent of the copper going to it was accounted for in various products.

When being operated without the shaft, on calcine having a temperature of 492° F., the reverberatory smelted 16,678 lb. per 24 hr. with an oil consumption of 797 gal. When running in connection with the roasting shaft, the furnace smelted 40,552 lb. cold, dry concentrates per 24 hr., with an oil consumption of 762 gal. When running this way, there were occasional delays due mainly to mechanical defects which were subsequently remedied or are capable of correction. The feed rate per 24 hr. of running time was 48,100 lb. Hence the furnace when operating with the shaft smelted approximately three times as much material in a

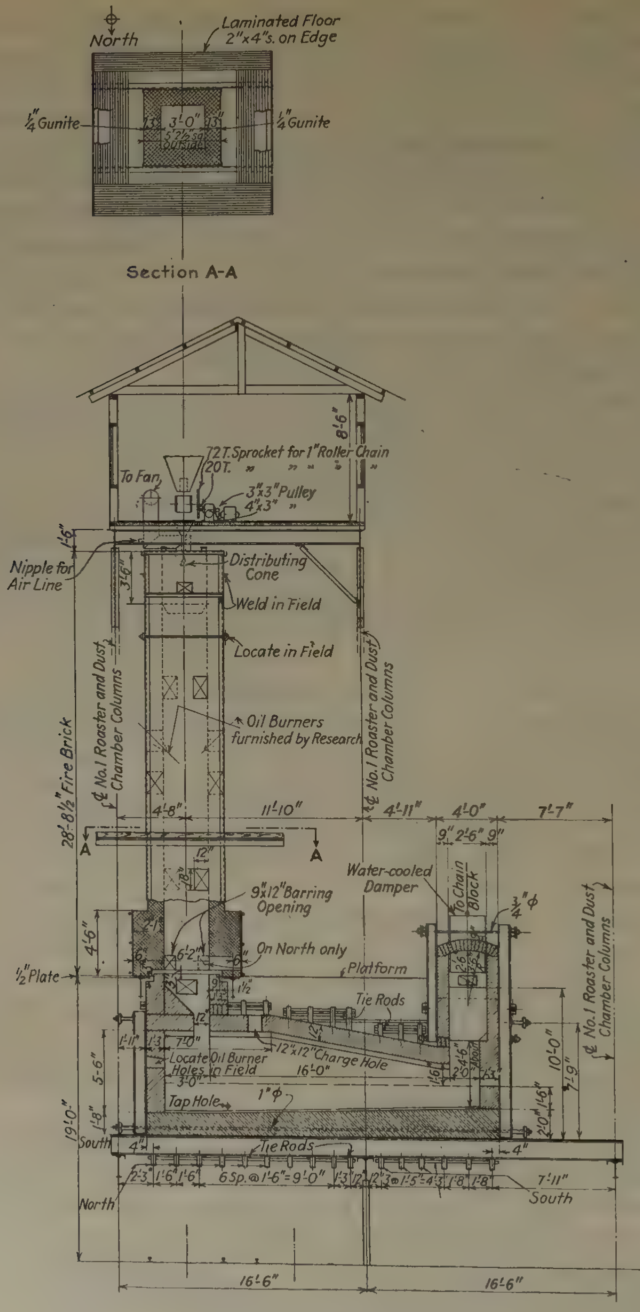

FIG. 1.—ROASTING SHAFT COMBINED WITH REVERBERATORY FURNACE.

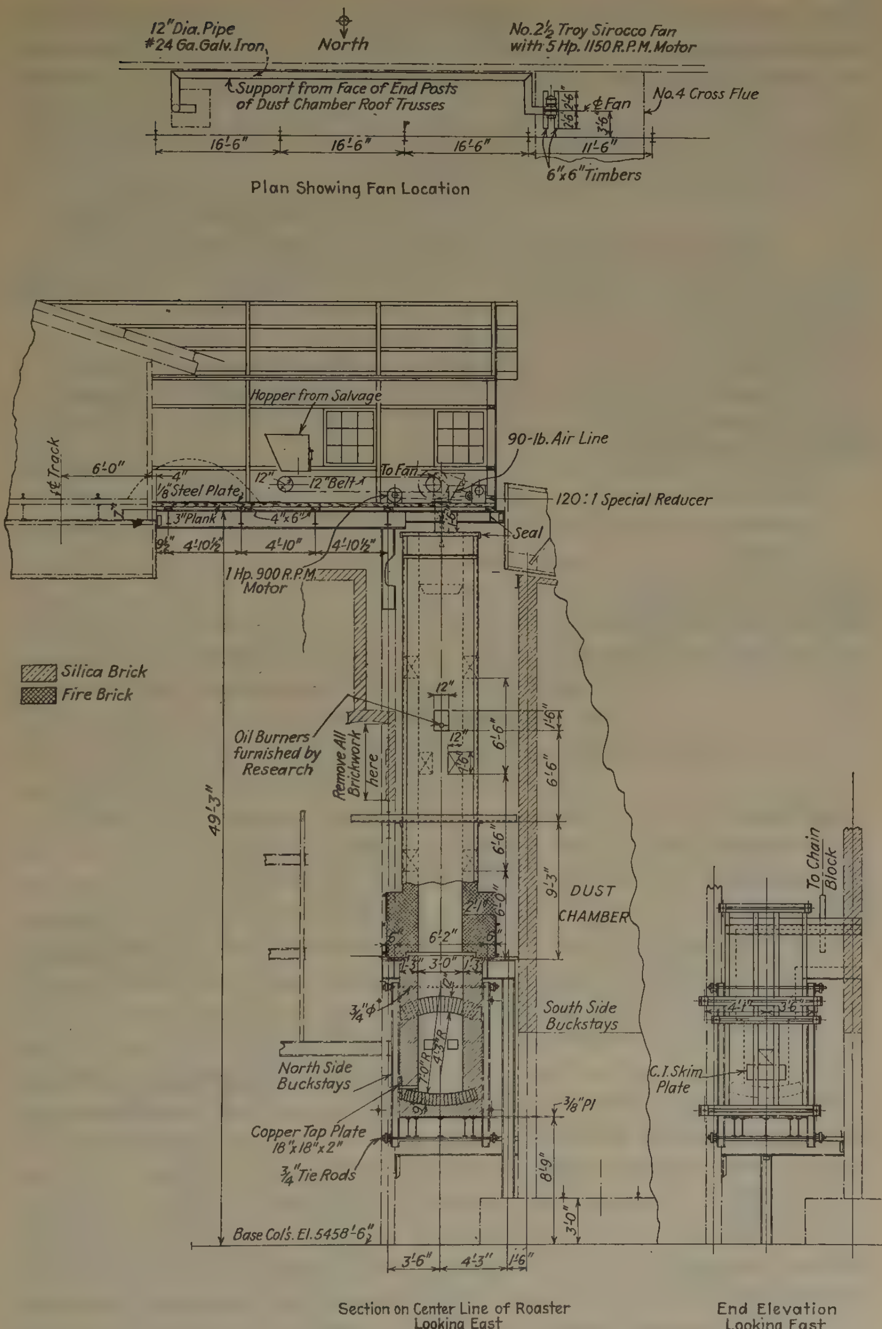

FIG. 1.—*Continued.*

given time, with the same consumption of oil, the oil consumption per ton of charge being about one-third what it was when the furnace operated without the shaft. After making due allowance for the calcination

TABLE 1.—*Performance of Reverberatory Alone versus Shaft Roaster plus Reverberatory*

	Reverberatory Alone	Shaft and Reverberatory
Kind of feed	Calcines	Concentrates
Duration of test, no. of shifts	30	38
Feed per 24 hr., lb.:		
Actual	16,678	40,552
Running	16,700	48,100
Feed temperature, deg. F	492	Cold
Shaft temperature, deg. F.:		
Top		580
Center		853
Bottom		1,059
Throat		1,470
Fan air per minute, cu. ft.		168
Injector air per minute, cu. ft.		43
Oil used per 24 hr., gal	797	762
Dust loss, percentage of total copper	0.92	2.36

ANALYSES, PER CENT

	H_2O	Cu	SiO_2	FeO	CaO	S	SO_2	O	CO_2
Concentrates	0.38	27.0	7.1	30.8		31.5			
Regular calcine		32.8	7.2	30.9	1.3	19.5			
Shaft calcine (*a*)		26.7	10.7	30.8	1.6	22.0			
Shaft calcine (*b*)		30.6		33.2		14.3			
Gas: shaft throat							11.7	3.7	
reverberatory flue								0.6	11.7

SHAFT DATA

	Connected with Reverberatory	Dry Feed, Shaft Alone	Moist Feed, Shaft Alone
Temperatures, deg. F.			
Top	580	886	277
Center	853	1,056	588
Bottom	1,059	1,220	1,103
Throat	1,470	1,508	1,420
Moisture, per cent	0.38	0.37	2.61
Feed per 24 hr., lb	40,552	80,911	87,655
Sulfur in feed, per cent	31.5	34.6	34.8
Sulfur in calcine, per cent	22.0	15.1	14.3
SO_2 in gases, per cent	11.7	10.1	

ratio (change in weight due to roasting) of the concentrates and making a reasonable allowance for delays, it may be said that the use of the shaft increased the capacity of the smelting furnace two and one-half times and decreased the fuel consumption per ton of charge by 60 per cent. These are the results obtained from consecutive runs of thirty 8-hr. shifts with the reverberatory alone, and thirty-eight 8-hr. shifts with the reverberatory operating in conjunction with the shaft.

This considerable increase in smelting capacity is due obviously to the high temperature at which the shaft calcine enters the smelting chamber (1470° F.) and may to a lesser extent be attributed to the hot gases from the roasting shaft.

When unhampered by the reverberatory, the capacity of the shaft when producing a calcine running about 15 per cent sulfur was more than 40 tons per 24 hr., or between 4.5 and 5 tons per square foot of cross-section in a shaft having an effective height of not more than 20 ft. The capacity is evidently a function of the volume of the roasting chamber, assuming that sufficient air to burn the sulfur and oxidize the iron is introduced with the concentrates. It seems reasonable to expect, therefore, that a shaft 6 ft. square, having an effective height of 30 ft., would roast around 250 tons per day when producing calcines containing 15 per cent sulfur. The plan that was considered for a larger furnace contemplated the equipment of a standard reverberatory with roasting shafts arranged along the sides. Three shafts approximately 6 ft. square by 30 ft. high on each side of the furnace might be expected to treat 1500 tons of concentrates per day.

It is not to be expected that a large reverberatory furnace would derive the same benefit from being connected with a shaft roaster that the small test furnace did, because the smelting efficiency of the large furnace is much greater. Any predictions as to the performance of a large furnace are necessarily pure guesswork. It may be taken for granted that the capacity would not be increased two and one-half times, but we are of the opinion that it would be increased at least one and one-half times and might be doubled, without any material increase in fuel consumption. By dropping the calcine along the furnace walls, protective embankments would be formed as usual and there would be a minimum of dusting.

Whether such a furnace would be practical and economical may be open to question. One of the obvious disadvantages is that the concentrates must be dried. Furthermore, the tendency is to produce concentrates of higher and higher grade, so that in many places the sulfur is only just sufficient to produce a matte of the desired copper content. Such concentrates, of course, must be fed directly into the reverberatory furnace without having any of their sulfur burned out. The process obviously is limited to the treatment of concentrates that are comparatively low grade and require roasting.

DISCUSSION

G. A. Guess, Toronto, Ont. (written discussion).—The increased smelting rate with the hot calcine was 1:2.5. If we give a value 0.18 to the mean specific heat of the calcine the difference in the heat content of the two calcines is 175 B.t.u. per pound. If we say that x B.t.u. added to the colder calcine will smelt it to slag and matte, $x - 175$ B.t.u. will be the requirement per pound for the hotter. If we have the same furnace-fuel conditions in each case and a smelting ratio of 1:2.5, we have

$$\frac{x}{x - 175} = 2.5 \text{ and } x = 292 \text{ B.t.u.}$$

the heat requirement to smelt one pound of the colder calcine. This is independent of the thermal efficiency of the furnace, but it assumes that that efficiency is the same in both cases. As a matter of theory the thermal efficiency when smelting the hotter calcine would be slightly lower, because there is a smaller temperature difference in this case between the calcine and the reverberatory gases.

This value, 292 B.t.u., seems too low, unless the matte fall is very high. Requirements for matte are much less than for slag. At 0.18 specific heat, 292 B.t.u. per pound would give a rise in temperature of 1630° F.; adding 492° F. gives 2122° F. There is no allowance for latent heat of fusion of the slag nor for the endothermic reactions in the calcine.

When we get at the B.t.u. requirement of a pound of calcine from a study of a thermal balance on a reverberatory, we are apt to conclude that 20 to 25 per cent of the heat in the fuel is actually in the resulting slag and matte. This may be called the thermal efficiency of the furnace. Taking a reverberatory burning 100 tons daily of pulverized coal (13,000 B.t.u.), it might be assumed capable of smelting 750 tons of the colder calcine:

1 lb. coal (13,000 B.t.u.) smelting 7½ lb. calcine
At 20 per cent efficiency 2600 B.t.u. in 7½ lb. 347 B.t.u. per pound
At 25 per cent efficiency 3250 B.t.u. in 7½ lb. 433 B.t.u. per pound

Taking the lower figure, 347 B.t.u. per pound, the hotter calcine would require 172 B.t.u. per pound and the smelting rate of the hotter calcine would be

$$347/172 \times 750 \text{ or } 1500 \text{ tons per day}$$

It would appear that the hot roaster gases cannot contribute to an increase of the smelting rate, since they are much colder than the reverberatory gas, and their low oxygen content would preclude the thought of their acting as preheated secondary air. It would appear from the analysis of the calcine that the matte fall is high, possibly 67 per cent, and that one pound of calcine will make about 0.27 lb. slag.

Forms of Copper Found in Reverberatory Slags

By Royal B. Jackman,* El Paso, Tex., and Carle R. Hayward,† Cambridge, Mass.

(New York Meeting, February, 1933)

Two comprehensive papers have appeared regarding the forms of copper that occur in smelter slags, one by Frank E. Lathe[1] and the other by C. G. Maier and G. D. Van Arsdale.[2] These authors comment on other published results and draw conclusions from their own investigations. The development of X-ray examination of minerals and improvement in high-power microscopic methods make it possible today to get additional evidence by means that were not available to earlier investigators. The present work was confined to reverberatory slags obtained from five different plants. Some definite conclusions were reached regarding the forms of copper present in these slags. In most cases these agree with some of the findings of Maier and Van Arsdale but new methods of attack were used in the investigation.

Conclusions of Previous Investigators

Various important conclusions reached by previous investigators are summarized in the following paragraphs, arranged chronologically.

William A. Heywood:[3] Slag losses increase with grade of matte but with the same grade of matte losses decrease as the acidity of the slag increases.

Lewis T. Wright:[4] Slag losses are only partly due to prills of matte. It is suggested that part of the copper is dissolved in the slag.

J. Parke Channing:[5] A considerable part of the copper enters the slag as silicate or oxide and not as sulfide.

C. A. Grabill:[6] Slag losses are both physical and chemical. Physical losses, due to poor settling, gas flotation and slight differences in specific gravity, are small under normal conditions. Chemical losses may be due to formation of oxides or to solution of metals and sulfides. The last is considered the most important.

John W. James:[7] The copper in converter slag is largely the oxide and as such enters the reverberatory slag when converter slag is added to the charge.

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† Associate Professor of Metallurgy, Massachusetts Institute of Technology.

[1] *Eng. & Min. Jnl.* (1915) **100**, 215.

[2] *Eng. & Min. Jnl.* (1919) **107**, 815.

[3] *Eng. & Min. Jnl.* (1904) **77**, 395.

[4] *Trans.* A.I.M.E. (1909) **40**, 492.

[5] *Trans.* A.I.M.E. (1910) **41**, 885.

[6] *Eng. & Min. Jnl.* (1910) **89**, 776.

[7] *Eng. & Min. Jnl.* (1914) **97**, 1114.

Frank E. Lathe:[8] Copper lost in slag decreases irregularly as silica content increases, and increases nearly uniformly with increase of copper in the matte. Considerable copper was found as oxide in the slag. This is believed to be the principal form in which the copper exists in slag. Serious loss as dissolved sulfide was not proved.

C. G. Maier and G. D. Van Arsdale:[9] Lathe's analytical methods are criticized. Little or no oxide copper is present. Most of the copper is in the form of sulfide, which may be either dissolved or in the form of visible globules. These globules are of different composition from the reverberatory matte, being apparently free from the iron sulfide phase. Copper has been identified only in the interior of sulfide inclusions.

F. E. Lathe:[10] Analytical methods used by various investigators are criticized and the presence of copper oxide or silicate maintained. Some dissolved copper is probably present.

Work Covered in Present Paper

Samples of reverberatory slag were obtained from five widely scattered smelters. Each weighed between 10 and 15 lb. and was in lump form when received. These were tested in the following ways:

1. Specimens were polished by the most improved methods and carefully examined at magnifications varying from 50 to 2000 diameters.
2. Chemical tests were applied to the polished surfaces to observe the reactions under the microscope.
3. Some of the slag was ground and treated by flotation. The concentrate was studied with X-rays and the diffraction patterns compared with patterns made from copper matte, bornite, chalcopyrite, chalcocite, cuprite and metallic copper.
4. A thin section of slag was examined by petrographic methods.
5. Leaching tests were made to see whether a substantial portion of the copper could be dissolved.

Results of Tests

Microscopic Work.—Photomicrographs are shown in Figs. 1 to 8, but the surfaces of the specimens themselves show differences in color that are not noticeable in the photographs. All specimens showed rounded particles easily recognized at 50 diameters as sulfides. It is a fair assumption that these resulted from matte inclusions. Scattered over the surface were numerous fine dots which at low magnification could not be identified but at high magnification their appearance approximated that of the larger particles. Both large and small particles are often associated with magnetite crystals, in some cases embedded in them. Although most of the copper particles are rounded, or globular in form, it is not unusual to find instances where they are quite irregular (Fig. 6). The form of copper seemed, in general, to be the same in all the specimens examined. The slag structure often varied, owing prob-

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[8] *Eng. & Min. Jnl.* (1915) **100**, 215, 263, 305

[9] *Eng. & Min. Jnl.* (1919) **107**, 815.

[10] *Eng. & Min. Jnl.* (1920) **110**, 1076.

ably to different rates of cooling, but the copper-bearing particles were scattered haphazardly throughout the mass.

Chemical Tests.—An attempt was made to identify copper minerals by using the methods prescribed by Farnham.[11] These tests indicated

FIG. 1.—FOUR MAIN CONSTITUENTS OF THE SLAG. × 100.
a. Copper sulfide; rounded white areas.
b. Crystals of magnetite; dendritic in this case.
c. Crystalline phase of siliceous portion; light gray.
d. Noncrystalline phase; darker gray.

FIG. 2.—FOUR CONSTITUENTS OF SLAG CLEARLY SHOWN. × 2000.
Sulfide particles and magnetite crystals are associated mainly with the dark noncrystalline phase, probably the last constituent to solidify.

the presence of bornite and covellite and possibly chalcopyrite. They gave no indication of the presence of metallic copper.

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[11] C. M. Farnham: Determination of the Opaque Minerals. New York, 1931. McGraw-Hill Book Co.

Flotation Tests and X-ray Examination of Concentrate.—Two samples of slag were treated by this method. The first was a granulated slag which was further ground wet to pass a 200-mesh screen. A small amount of sodium carbonate was added during the grinding and pine oil with potassium ethyl xanthate was used as the flotation reagent.

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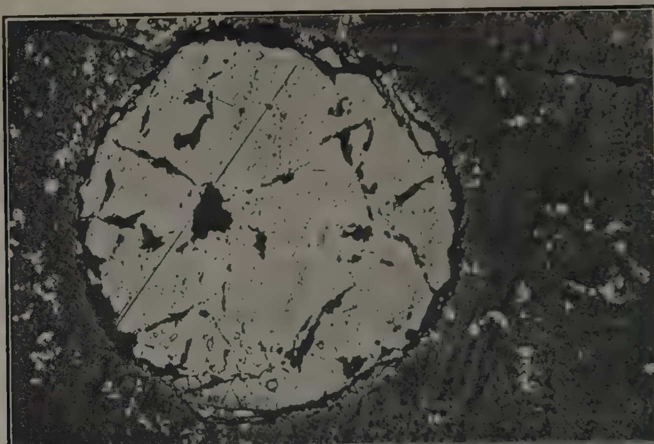

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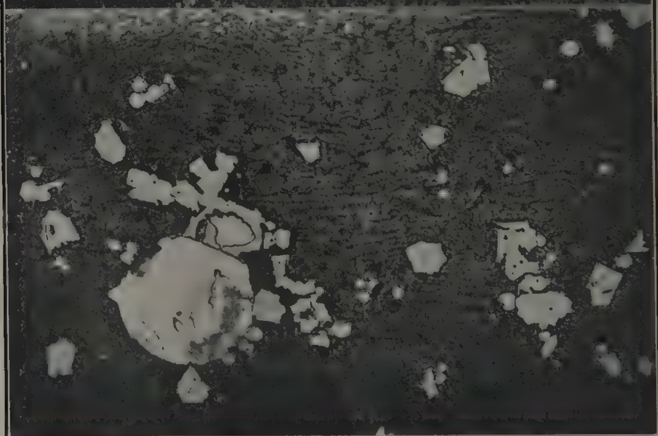

FIG. 3.—LARGE INCLUSION THAT WAS DEFINITELY IDENTIFIED AS SIMILAR TO BORNITE. × 100.
FIG. 4.—SMALL INCLUSION SIMILAR TO THAT IN FIG. 3. ALSO, SULFIDE PARTICLE EMBEDDED IN MAGNETITE CRYSTAL. × 500.

The concentrate from this test contained 3.59 per cent copper and the recovery was 32.8 per cent. There was considerable colloidal material present, indicating too much grinding, so in treating the second slag care was taken to avoid this trouble. No Na_2CO_3 was added, the pulp having a pH value of 9.8 after grinding. This test, with the same flotation reagents as those used in the first, gave two concentrates with 17.7

and 5.62 per cent copper respectively and a recovery of 48.2 per cent. Further cleaning of the richer product gave a concentrate with 45 per cent copper, which was used for X-ray examination. A study of this concentrate with a binocular microscope indicated that it was a mixture

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FIG. 5.—ROUND SULFIDE PARTICLES AND NUMEROUS SMALLER ONES HAVING THE SAME APPEARANCE AS THOSE IN FIGS. 3 AND 4, PROVING SIMILARITY OF SMALL AND LARGE PARTICLES. × 2000.

FIG. 6.—IRREGULAR MASS OF SULFIDE AND MANY SMALLER PARTICLES. × 500.

of bornite with possibly a little chalcopyrite and a gangue composed of silicates and magnetite.

The X-ray examination of the concentrate was carried out with a Siegbahn-Hadding gas-type X-ray tube with a chromium target. It was operated at about 35 kilovolts and 10 milliamperes. The powder method was employed, using a Westgren focusing camera. Diffraction patterns were obtained for the slag concentrate, bornite, chalcocite,

chalcopyrite, cuprite, copper reverberatory matte and metallic copper. Positive prints are shown in Fig. 9.

Examination of these patterns will show that the lines for bornite correspond closely to certain lines present in the slag concentrate pattern. The chalcocite pattern is very similar to the bornite pattern, the only difference being that the lines do not exactly concur. This is due to the

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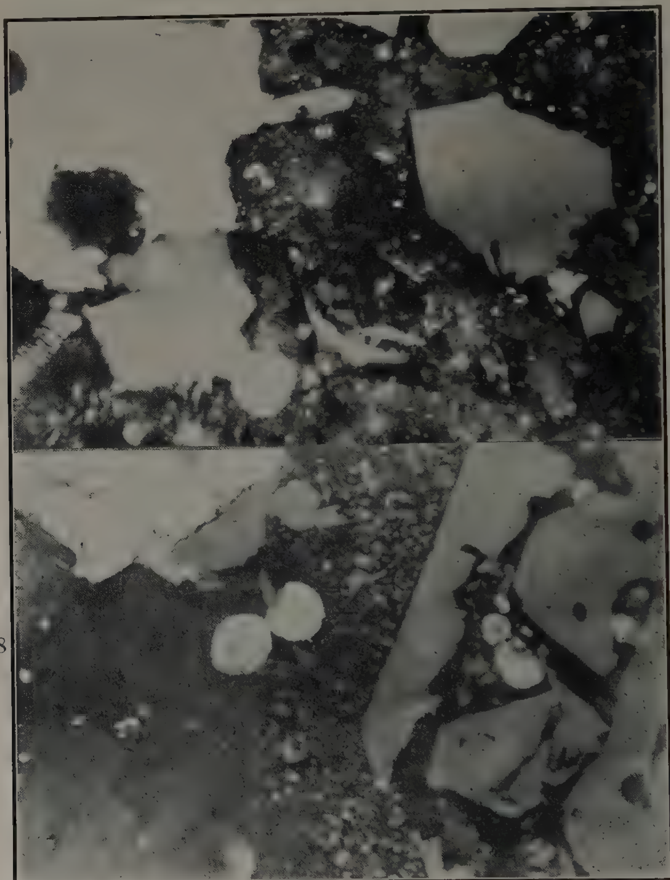


FIG. 7.—MAGNETITE AND SULFIDE PARTICLES IN CLOSE ASSOCIATION. × 2000.
FIG. 8.—IRREGULAR AND ROUND PARTICLES OF SULFIDE. × 2000.

fact that bornite is a copper sulfide containing some iron in solid solution, while chalcocite is pure copper sulfide. The presence of the iron changes the position of the lines with respect to the origin, but otherwise does not affect the pattern.

This observation indicates that the copper in the slag concentrate is in a form similar to bornite; that is, a copper sulfide containing varying amounts of iron or iron sulfide in solid solution. Several lines for the

copper reverberatory matte appear to correspond to some of those in the slag concentrate pattern, and it is assumed that some substance similar to matte is also present in the slag concentrate. Since no similarity can be detected between the pattern for the slag concentrate and those for chalcopyrite, cuprite and metallic copper, these substances are considered as being absent in the concentrate.

Examination of Thin Section.—Only one specimen was examined by this method, and the results were disappointing. The slag consisted mainly of two portions, one a transparent yellowish crystalline mass

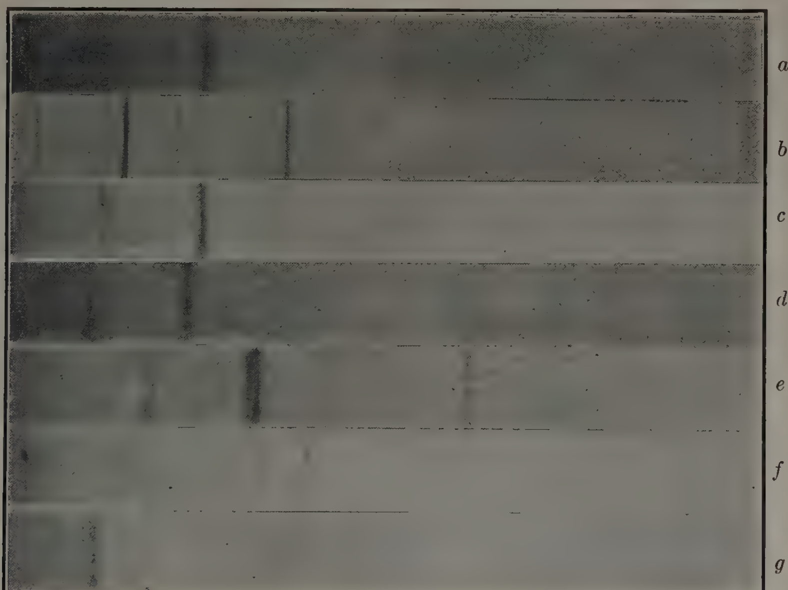


FIG. 9.—DIFFRACTION PATTERNS OF SLAG CONCENTRATE AND CONSTITUENT MINERALS.
a. Slag concentrate. *b.* Copper. *c.* Bornite. *d.* Chalcocite. *e.* Chalcopyrite. *f.* Copper reverberatory matte. *g.* Cuprite.

and the other an opaque, dark crystalline matrix: Magnetite was readily recognized but nothing that resembled the sulfide globule could be distinguished.

Leaching Tests.—This work was carried out in the metallurgical laboratories of the Massachusetts Institute of Technology by E. M. Tittman and L. W. Johnston several years ago, when the price of copper was higher than at present and when a copper content of 0.6 per cent in reverberatory slag meant a considerable daily loss. A summary of their results is presented here as a corollary to the facts brought out in this paper.

The material used was a granulated reverberatory slag containing 0.6 per cent copper. Leaching was carried out by adding the reagent

to the slag samples and grinding for various lengths of time in a porcelain mill with porcelain pebbles. Tests with dilute sulfuric acid gave, as was to be expected, a low extraction. Further work was done with ammonia and ammonium sulfate, or ammonium carbonate, with the addition of various quantities of hydrogen peroxide.

Tittman obtained his best results with a leach solution consisting of water with 5 per cent ammonium sulfate, 5 per cent ammonium hydroxide and 5 per cent hydrogen peroxide. In one test he obtained an extraction of 91.9 per cent after grinding 20 hr. Johnston attempted to duplicate Tittman's work but, with the same solutions, the best extraction he could get was 73.4 per cent in 114 hr. He then made some tests substituting ammonium carbonate for ammonium sulfate. The best result with this solution was 64.4 per cent in 73¾ hr. On all these tests the product was extremely fine after grinding and gave considerable difficulty in filtering and washing.

Discussion of Results

From the evidence produced in the various tests used in the present research, and evidence presented by previous investigators, it seems to be definitely established that most of the copper contained in reverberatory slag is in the form of sulfide. Most of this sulfide is in the form of pellets of irregular shape, but some sulfide may be dissolved in the slag. Metallic copper may be present at times, but it is of minor importance.

By comparing the amount of copper analytically determined in the slag and the amount visible in the form of sulfide, it is evident that very little, if any, remains in any other form. No evidence was found of the presence of copper silicates and, although proof of their absence was also not obtained, it is probable that those who claim that such silicates are not present in normal slags are correct.

The presence of numerous extremely fine particles of sulfide suggests that when the slag is in the furnace it may hold considerable sulfide in solution, which may separate during cooling.

It is probable that one of the most important sources of copper in the slag is the dust that settles on the surface. These particles are so fine that they would not readily agglomerate and settle out. The oxides are readily dissolved and the sulfides either dissolved or fused into globules. These globules would not necessarily have the composition of the furnace matte but undoubtedly would approach the composition of bornite as indicated in the X-ray examination. Any copper oxides present would undoubtedly react with the sulfides and the copper would be taken up in these globules.

Maier and Van Arsdale have shown that considerable matte may be floated into the slag by ascending gas bubbles. These particles undoubtedly would have a slightly different composition from those produced

from ore dust, but as these authors have pointed out, the composition is also different from matte, probably because the reaction between the matte and the oxides in the slag cause an oxidation of FeS. The final composition of these particles will depend on the extent to which the reaction goes. In some cases it may proceed to the complete removal of iron and sulfur, leaving metallic copper.

This reaction is dependent to a considerable extent on the temperature. Reverberatory matte at its melting point will not react with magnetite, but as the temperature is raised, a point is reached where the reaction is violent. The latter temperature is a variable depending on matte composition, amount and condition of Fe_3O_4, concentration of SO_2 and pressure. The last item depends on depth below the surface of the bath. In a reverberatory furnace, one area may not be reacting while another is reacting violently. Near the bottom of the furnace the matte is cool and has little tendency to react with magnetite. Here it may be saturated with magnetite and actually depositing it from solution. At or near the contact line between slag and matte there are undoubtedly areas in the furnace where the reaction is taking place and particles of matte are driven into the slag. The temperature continues to rise as the surface of the slag is approached, so the violence and completeness of the reaction will increase if the gas evolution continues to carry the particle upward. The final result is a particle in the slag which may vary in composition between matte and metallic copper.

Where copper silicates occur in reverberatory slag, they are probably the result of incomplete digestion of converter slag.

The removal of copper from slag by agitating it with pyrite, or by adding pyrite near the skimming end of the furnace, is probably due more to physical and mechanical action than to chemical action. The relatively large amount of low-grade matte produced from the fusion of the pyrite collects the dispersed particles of copper sulfides and removes them as the matte settles from the slag.

Either the above method, concentration by flotation, or leaching are available for cleaning slags. At present they are only of academic interest but at some distant date one of them may be practiced to conserve copper resources.

DISCUSSION

(Frank G. Breyer presiding)

C. G. Maier, Berkeley, Calif. (written discussion).—This paper contains two items in connection with the investigation of copper reverberatory slags that represent a definite advance as compared to older work. The first of these is the better technique in the taking of photomicrographs. This improvement, unfortunately, has not added appreciably to knowledge of the subject, except as it indicates the presence of minute sulfide inclusions. Since such minute inclusions cannot be recognized with

entire definiteness under the highest magnifications, it is perhaps doubtful whether the evidence here is conclusive.

A second interesting contribution of these authors is their attempt to use X-ray methods to determine the nature of the sulfide inclusions. Unfortunately, the fact that they could make these X-ray tests only upon flotation concentrates representing a recovery of approximately 50 per cent greatly decreases the value of their findings. That the larger prills in the slag are essentially cuprous sulfide does not require proof by such a ponderous method as the use of X-rays, since the fact is definitely ascertainable both by visual microscopic observation and by microchemical tests. As the conclusion of the X-ray tests is limited to the 50 per cent that was recovered by flotation, the matter of the remaining half of the copper content of the slag is still in a doubtful condition.

The statement that the presence of numerous fine particles of sulfide suggests the possibility of sulfide in solution, which separates during cooling, seems to me unjustified. It does not appear that sulfide separated in this way could be visual, even under the highest magnifications used by the authors. At the most the solubility in the slag of cuprous sulfide must be less than about 0.3 per cent, which corresponds to a rather dilute solution. Such a dilute solution, in separating out a relatively insoluble constituent upon cooling in an environment of high viscosity, would certainly result in such minute particles of sulfide as to be undetectable by microscopic means, and, moreover, uniformly distributed throughout the slag. The evidence for the presence of copper sulfide in solution in slags still remains almost entirely indirect.

An obvious interpretation of Johnston's leaching tests is that some 65 to 75 per cent of the copper content is of such particle size as to be freed by grinding, but that roughly one-third of the copper content is in a form not readily attacked by the reagents, and presumably may be very finely divided, or in solid solution. Flotation tests lead to a somewhat similar conclusion. The fact that the authors have used the latest refinements of microscopic technic and have further attempted to utilize X-ray methods, and in the end, succeeded only in reaching a result so equivocal as to fail to convince the more dogmatic of copper metallurgists, should ultimately discourage those in the future who may wish to study copper slags by these methods.

The results have brought the question to a point where all that remains is a purely physical-chemical problem: What is the solubility of cuprous sulfide in slag? This is a problem not incapable of answer by physical-chemical means.

A. Zentner, San Francisco, Calif.—Some years ago I examined Anaconda slag that had been granulated by pouring into the water launders, and found numerous apparent shots of copper in the samples. I took these shots, crushed some and cut others with a knife. They were matte coated with a very thin film of metallic copper. Odors of H_2S and SO_2 at the slag launders were probably due to superheated steam reacting with the matte to produce H_2S and Cu_2O, the latter in turn reacting with matte to form SO_2 and the films of copper. It is therefore possible that metallic copper was present in the first set of samples, and was not due to the rouge used in dressing the slides.

F. F. Colcord, New York, N. Y.—What bearing does acidity of slag have on its copper content?

C. R. Hayward.—We did not go into it. Lathe has shown that there is a relation. The question of visibility of precipitated sulfides is anyone's guess. Two thousand diameters is a large magnification and the samples were carefully prepared.

S. Rolle, New York, N. Y.—In my opinion, Mr. Colcord has started the proper discussion, as so far we lack the quantitative data necessary to a proper approach. We cannot weigh the evidence till we have such data.

F. E. Lathe, Ottawa, Ont.—If, as some claim, molten slag contains considerable copper as dissolved sulfide, much of this may be expected to separate on cooling and be found in the solid slag as sulfide particles. The presence of dissolved sulfide has probably not been proved, although various workers have referred to it in published papers. For example, Wanjukoff, an early worker who wrote a long article in German on this subject, is often quoted as authority for the presence of dissolved sulfide. Those who take the trouble to read his paper, however, will find that he tried to devise a method of determining copper oxide in slag, and failing to do so applied the term dissolved sulfide to whatever copper was not present in globules of matte. While at the Granby I got strong evidence of the presence of oxidized copper, although some copper was always present as matte particles. The nature of the charge was correlated with the condition of the copper in the slag; thus when oxidized ore was on the charge the oxide content of the slag, as determined by the old sulfuric acid method, always increased. The difference between the gold-copper ratios of the matte and of the slag furnished a rough check on the method, on the assumption that only matte particles carry gold into the slag. Later, improved analytical methods on blast-furnace, reverberatory-furnace and converter slags from a number of different smelters indicated the presence of oxide copper in widely varying amounts, giving a good correlation between the laboratory and practice. There was much oxide copper in those slags derived in part at least from oxidized ores, whereas those produced under more strongly reducing conditions contained relatively less oxide.

I recently had the opportunity to study the gold-copper ratios of mattes and slags relatively high in gold and have again found a much smaller proportion of gold passing into the slag; accounted for, in my opinion, by the fact that gold is not oxidized, as is copper. While the evidence is not yet fully conclusive, it is my belief that suspended matte particles account for some 25 to 50 per cent of the copper in most slags. The condition of the remainder may be open to doubt, but it appears that most of it is present as copper oxide or silicate. I have never observed metallic copper in slag, but have seen both metallic copper and metallic iron in mattes. These metals are produced under very strongly reducing conditions.

C. R. Hayward (written discussion).—Mr. Maier has made an excellent analysis of the conclusions presented in the paper and most of his points are well taken. The authors believe that the flotation concentrate, while not necessarily containing all the free sulfide particles, should at least have represented in them some part of all the forms of sulfide present. The presence of several forms leads to interesting speculation as to their origins, but such speculation is out of place here.

In spite of Mr. Maier's opinion to the contrary, we still believe it is entirely possible to have particles of sulfide, precipitated from solution in the slag, of sufficient size to be seen under high-power magnification. His final suggestion regarding the importance of determining the solubility of sulfides in slag is excellent. If this could be done it would throw much light on the subject of copper losses in slags.

Mr. Lathe's opinions are entitled to respect, even though we may not agree with him. No evidence of oxidized copper was obtained in studying the slags discussed in this paper, but we must admit that negative evidence is quite different from positive evidence. The numerous particles of sulfide, large and small, visible in the specimens examined, gave a distinct impression that all, or practically all, of the copper was represented in this form, but obviously no definite statement can be made regarding it. With variations in charge, slag composition, furnace conditions, rates of cooling of slag samples, etc., it is nearly impossible to get sufficient evidence to make the results of any research unqualifiedly convincing. The authors of this paper make no such claims. The results are presented as a contribution to the general knowledge of the subject.

Development of Copper Converting at Butte and Anaconda

By William Kelly*, Anaconda, Mont., and Frederick Laist,† New York, N. Y.

(New York Meeting, February, 1934)

The slow, tedious and costly method of reducing copper matte to metallic copper in the reverberatory furnace, commonly known as the Welsh process, was displaced by the rapid and inexpensive converter process about fifty years ago. The art of refining copper was for many generations a carefully guarded secret among the Welsh furnacemen and

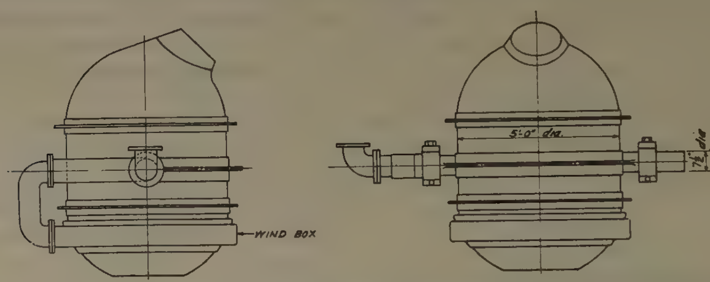


Fig. 1.—Parrot converter. *(From Peters: Practice of Copper Smelting, 479. By permission of McGraw Hill Book Co.)*

was handed down among them from father to son. An attempt was made to apply it at Butte when the smelting of copper ore first started there but it proved too difficult and expensive under local conditions, and until the introduction of the converter process the entire output of copper was shipped to Swansea in the form of high-grade (60 per cent) matte.

Before the construction of the Manhes converter, attempts had been made to reduce copper matte in the bessemer steel converter, but these failed, chiefly because the air was blown through the bottom of the vessel. The air holes became clogged with chilled copper as soon as the reduction started. Manhes partially overcame this difficulty by placing the tuyeres in the side of the vessel, some distance above the bottom, thus allowing space beneath them for the copper to accumulate. Nevertheless the first attempts to operate such a converter at Butte were not successful, as no means had been provided for clearing away obstructions, which formed to some extent in front of the tuyeres. This difficulty was overcome by drilling holes in the outside wall of the air chamber directly opposite the holes in the shell and lining, so that these could be punched

* General Foreman, Anaconda Reduction Works.
† General Metallurgical Manager, Anaconda Copper Mining Co.

with a steel rod while the converter was in operation, the outer holes being kept closed usually with wooden plugs.

Dr. Peters, who was at that time manager of the Parrot smelter at Butte, recommended to the owners the purchase of patent rights on the

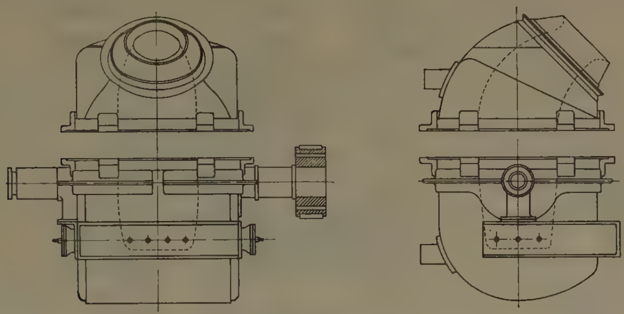


FIG. 2.—STALMANN TYPE OF CONVERTER. (*From Borchers: Metallurgy*, 98–99. *John Wiley & Sons.*)

Manhes converter, and in 1884 an agreement was signed between the interested parties. The first copper converter in the United States was installed at the Parrot works during this year and commenced operating in September. It was a small, egg-shaped vessel, made up of three

FIG. 3.—HIXON CONVERTERS, OLD WORKS, 1893.

sections bolted together. The tuyeres encircled the lower section and pointed toward the center. The stands were constructed so as not to interfere with the punching of the tuyeres and the converter was tilted, at first by means of a windlass, later by an arrangement of pulleys and

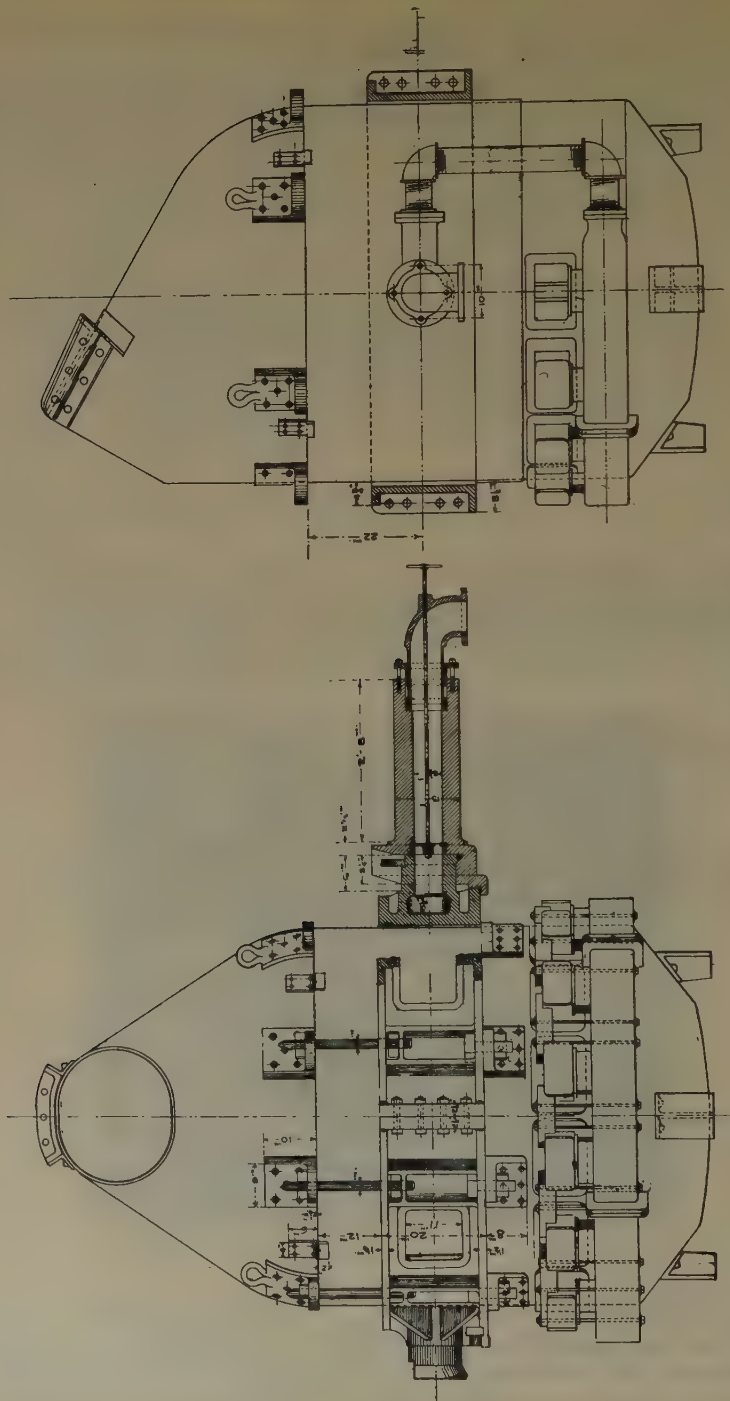


Fig. 4.—Converter, Anaconda, 1897.

belts. The matte was tapped directly from the settler of a 70-ton blast furnace situated about 8 ft. above the converter floor level.

This experimental installation was followed by three converter stands and a small cupola for remelting matte from the reverberatories, and this little plant produced from 1¼ to 1½ million pounds of copper per month

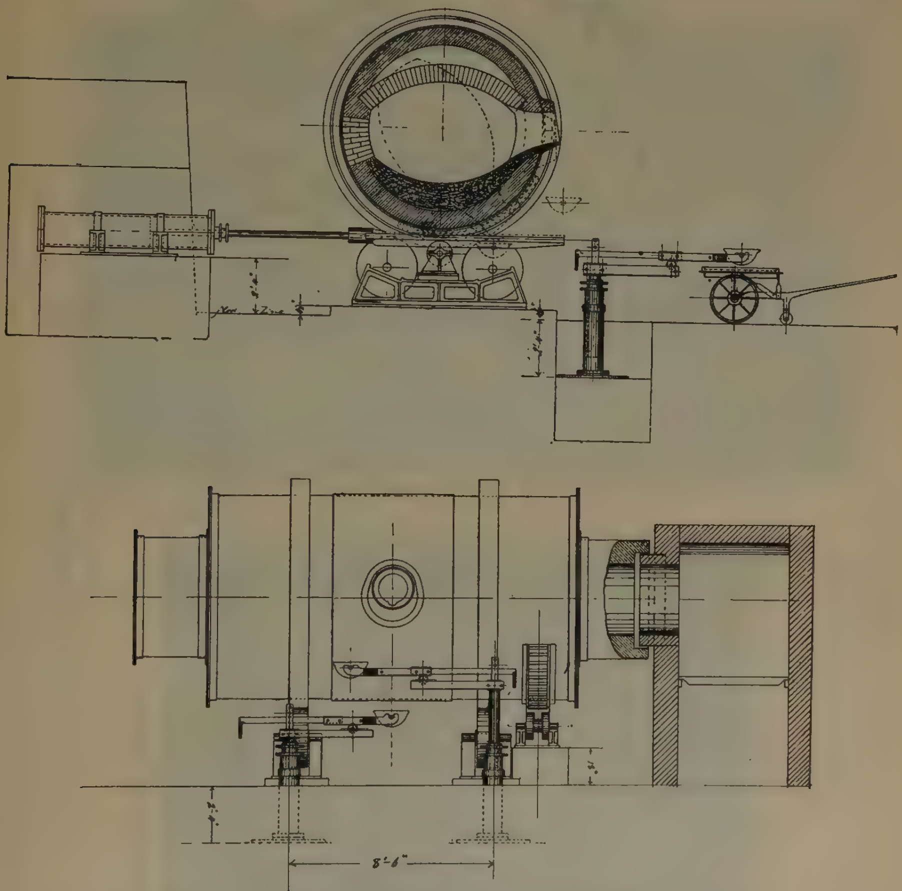


FIG. 5.—TILTING FURNACE, ANACONDA, 1897.

with two converters operating. These early converters were lined by hand with a wet mixture of crushed silica and fireclay. The bottom was lined first, after which the top sections were bolted on and lined with the same mixture made up into balls, which were passed to a man on the inside who pounded them into position. The charge of matte to a newly lined shell was about 2 tons, which increased to 4 or 5 tons by the time the lining was nearly worn out.

In 1891 a small converter was built at Anaconda by Otto Stahlman, for experimental purposes only, which held about one ton of matte.

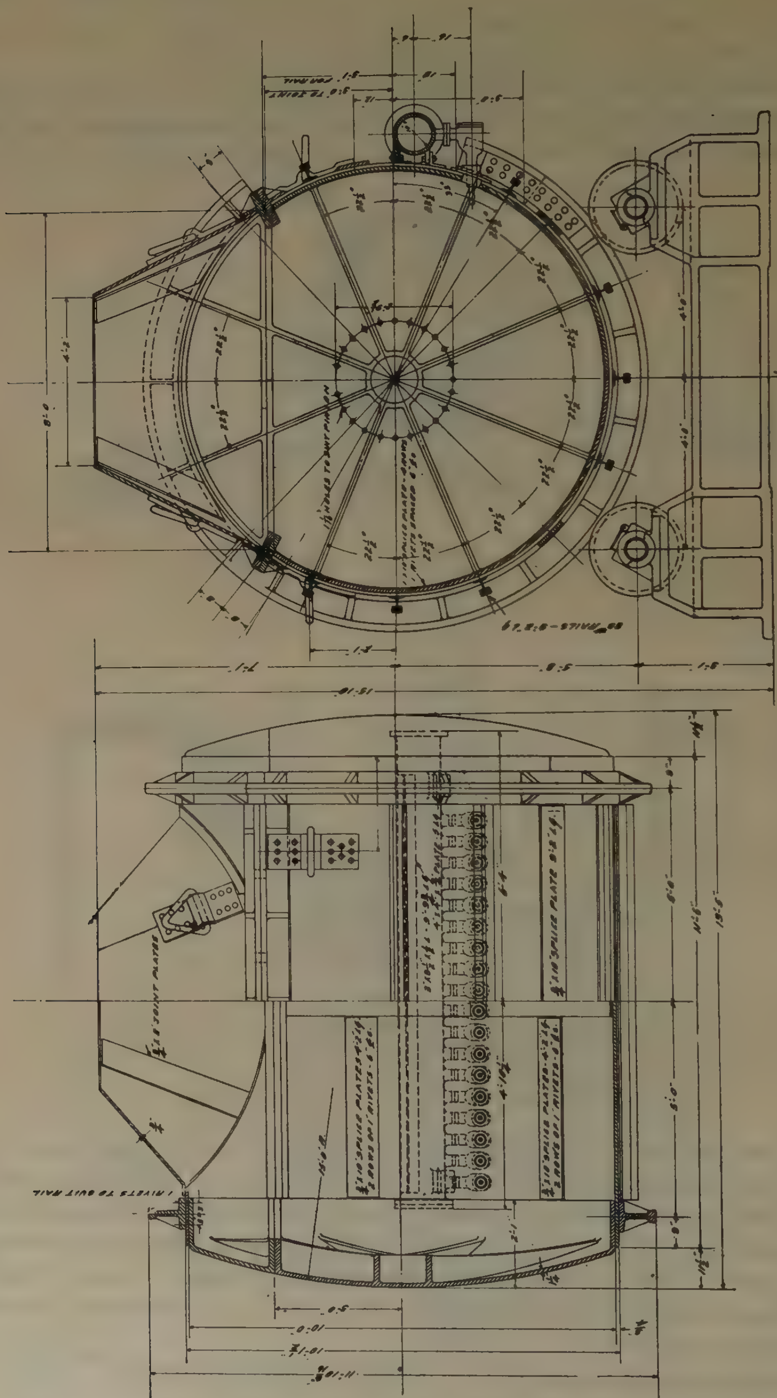


FIG. 6.—CONVERTER, ANACONDA, 1902.

It was followed by a plant consisting of five square converters that each held about 5 tons of charge after the lining was partially corroded. Except for being square in cross-section, instead of round, these were similar to the Parrot converters in all respects. There were tuyeres in the back and sides. They were tilted by belts and pulleys. There were

Fig. 7.—Converter floor, New Works, 1904.

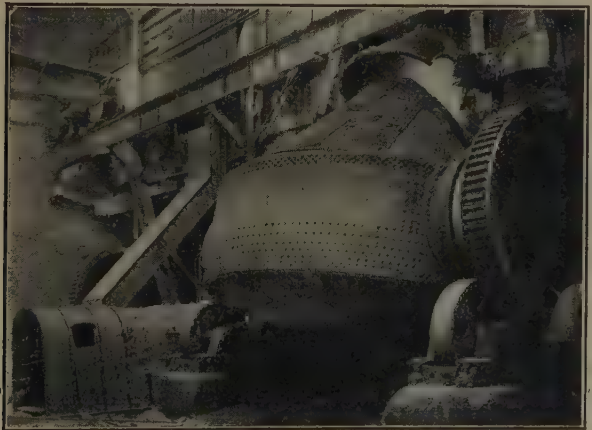


Fig. 8.—Great Falls type of converter.

five Fraser and Chalmers compressors, one for each converter. Later an air receiver was installed and all of the compressors and converters were connected to it. Slag was poured into small pots and wheeled to the dump for subsequent reclamation and treatment in a blast furnace. Copper was cast into 200-lb. slabs. The plant was subsequently enlarged by seven stands, making a total of twelve converters.

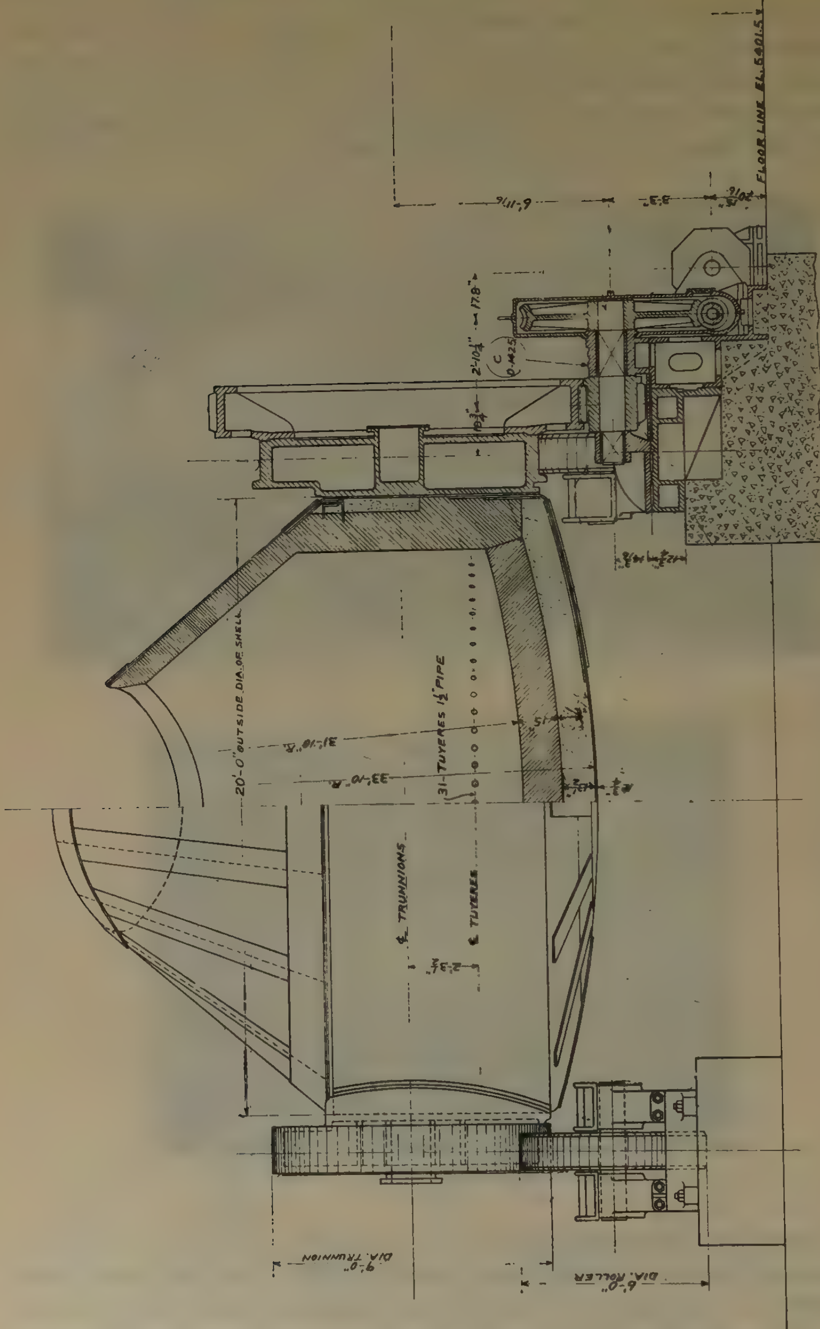


FIG. 9.—TWENTY-FOOT CONVERTER, GREAT FALLS TYPE.

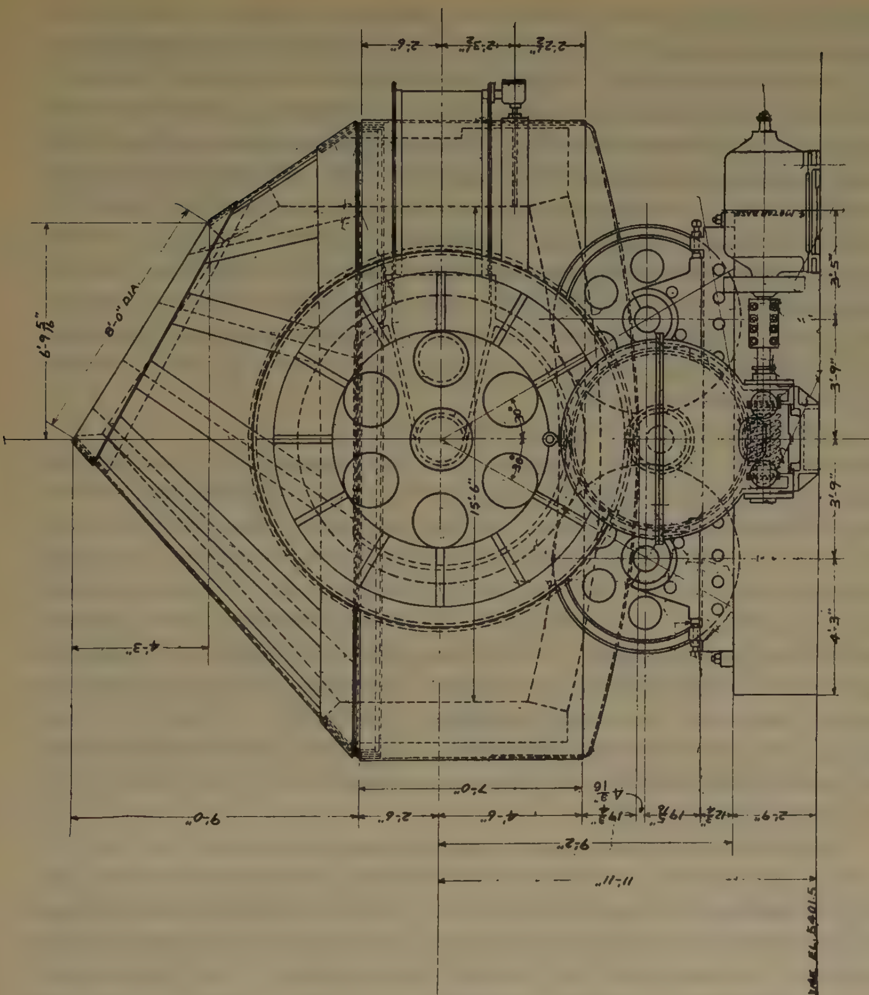


FIG. 9.—*Continued.*

The lining cost was a considerable item of expense and about 1893, at Marcus Daly's instigation, an extended investigation was made under the direction of Mr. Hixon with a view to developing a more durable lining, the necessary silica being supplied through the converter mouth. Many different lining materials were tried but without success. During the course of these trials, water pipes were embedded in the lining at a depth of about eight inches. As soon as they became exposed, copper built up on them and thereafter the charge cooled so rapidly that the blow could not be finished. Magnesite was not tried as a lining material.

Under Hixon's direction a new plant was built at the Lower Works and went into commission in 1894. The plant contained eight round

upright converters, 10 ft. high by 6 ft. diameter. Four were added later, making a total of twelve. There were also six cupolas for remelting, one for every two converters. These converters were connected through a flue and dust chamber with a stack on the hillside. They were tilted hydraulically and were served by a 25-ton overhead crane. There was a lining department, supplied with crushing facilities and tamping machines, to which the converters were taken by means of the crane, when they required relining. This was quite an improvement over the old method of relining by hand with the converter in place. The lining was tamped around a core, which could be withdrawn in sections, and lasted considerably longer than when put in by hand. These converters held from 8 to 9 tons of matte after partial corrosion, and produced in all about 10,000,000 lb. of copper per month.

At this plant an attempt was made to use a basic lining. A shell was lined with magnesite brick and put into operation. The converter performed poorly, however, perhaps owing to its small size and the fact that the lining was 18 in. thick. After a few trials a suggestion was made that the thickness of the lining be reduced to 9 in., but this was not done, and the work was dropped. It is quite possible that if this suggestion had been followed, and if a little more persistence had been shown, success might have been attained and the basic converter might have made its appearance on the scene 15 years sooner than it did. It has been demonstrated since then that small size alone is no bar to the use of a basic lining. Bancroft Gore operated a small basic-lined, hand-tilted converter at Gatico, Chile, the size of which was considerably less than the old converters at Anaconda.

One of the important improvements made about this time was the Dibblie ball tuyere. It replaced the wooden plugs that were first used and the sliding steel caps that followed. Both of these means of closing the holes in the wind box were clumsy and wasteful of air because they were hard to keep tight and difficult to manipulate. The Dibblie ball-valve arrangement enabled punching to be done with a minimum loss of time and air. Later individual tuyeres replaced the wind box arrangement.

In 1902 a new converter plant was built in connection with the Washoe smelter at Anaconda, containing 13 stands with converters of the horizontal type, 10 ft. diameter by 13 ft. 6 in. long. They were tilted hydraulically, had eighteen 1-in. tuyeres and held from 15 to 20 tons of matte. The lining was a mixture of cuprous slimes from the concentrator and siliceous ore fines. It was tamped into place in the usual manner, the shell being transferred from its stand to the lining department and back again by a 60-ton crane. A lining lasted from four to six charges. Its life could be considerably prolonged by feeding in siliceous ore but the practice was not adopted to any great extent.

When the Peirce-Smith converters were developed at Garfield, the shells in the Anaconda plant were lined with magnesite brick and operated successfully as basic-lined converters until they were replaced by the large upright converters of Great Falls design, which are still in use. Thirteen small converters produced about 18,000,000 lb. of copper per month, whereas seven large converters produce over 27,000,000 lb. per month from matte averaging about 42 per cent copper, including all lost time. They have produced 30,000,000 lb. per month from higher grade matte such as can readily be made when circumstances render it desirable to do so. A converter holds about 75 tons of matte at a charge and can produce about 85 tons of copper per 24 hr. from 45 per cent matte.

Developments in Converting Lead and Copper Matte at Tooele

By B. L. Sackett,* Tooele, Utah

(New York Meeting, February, 1934)

The converting of lead matte is not a general practice at lead smelters, therefore a description of the methods used and developments made during the past 20 years in converting both lead and copper matte at the Tooele plant of the International Smelting Co. in Utah may be of interest.

The Tooele converting plant consists of five electrically operated stands in which are used 96 by 150-in. horizontal, barrel-type converters each of about 10 tons capacity. The main converter building is served by a 60-ton electric traveling crane and the casting shed by a 30-ton crane. Copper matte is tapped from near-by reverberatory furnaces into launders that deliver into 8-ton ladles on the converter floor. Lead matte is brought from the blast furnaces, about ¼ mile away, in 5-ton ladles mounted on trucks, hauled by electric locomotives. As the Tooele copper plant has never been a producer of much copper, and has been irregular in its operation, the original converting plant with its small converters has been adequate, although decidedly out of step with large, modern plants. Tonnage figures in this paper are for comparison with each other only, as they are not at all in line with prevailing tonnages at larger plants today.

The reasons for the changes made in converting practice at Tooele, as well as the improvements in the results obtained as these changes were perfected, are given in the following pages.

Lead-matte Converting, 1914 to 1927

In 1914, Oscar M. Kuchs wrote a paper for the Transactions,[1] describing in detail the lead-matte converting methods at Tooele up to that time. The method perfected only a few months before Mr. Kuchs' paper was written consisted, briefly, of blowing the lead matte alone, without the addition of siliceous fluxing ores, to a point where the lead and zinc were largely eliminated by volatilization. During this period

* General Superintendent, International Smelting Co.

[1] O. M. Kuchs: Lead-matte Converting at Tooele. *Trans.* A.I.M.E. (1915) **49**, 579.

of blowing it was sometimes possible to skim from the converter a small amount of high-iron slag, but usually this was not done; instead, the entire remaining charge, very high in iron and copper, was transferred at the end of this period to another converter containing copper matte, where it was blown, with the addition of siliceous ore, to blister copper. The fume in the gases from the lead converters was recovered in a baghouse containing 960 woolen bags, 18 in. in diameter by 30 ft. in length, served by a No. 20 Sirocco fan of 80,000 cu. ft. capacity. The bags were cleaned by the reverse current method of shaking.

For a period of about 13 years, or until the spring of 1927, this method of converting lead matte was used with generally satisfactory results. The one outstanding difficulty during these years was the high temperature too often existing in the gases going to the baghouse, which caused considerable interruption in blowing the lead matte. The only time at which two lead converters could be up together was when one was just finishing and the other just starting the blowing of a charge, as both of these periods of blowing were comparatively cool. This, of course, was largely caused by insufficient length of flues, or other gas-cooling provisions, between the converters and the baghouse.

During most of this time the copper-converter gases were handled through a separate flue system where they were cooled and humidified with water sprays in an acidproof brick chamber and subsequently treated in a pipe-type Cottrell precipitator.

With this kind of lead-matte converting, several points of particular interest should be brought out:

1. In blowing without the addition of silica there was a period, about two-thirds through the blow, when the converter gases became very hot, making it difficult, even when blowing only one converter, to keep the temperature of gases cool enough (215° F.) to be safely handled through the baghouse. This accentuated the hot-gas troubles due to mechanical limitations, causing delays in operation.

2. The tonnage treated per converter day in both lead and copper converters was considerably lower than it has been during the past few years, since lead and copper matte have been blown together and relatively large amounts of siliceous ores used. (The total tonnage of hot and cold matte, "dope" and siliceous ore treated in 1926 was about 110 tons per converter day, whereas in 1931 this was increased to 143 tons, or an increase of 30 per cent. The amount of siliceous ore treated per average lead-plus-copper converter day increased from 10 to 25 tons, which in itself accounts for nearly one-half of the total increase.) The reasons for this low tonnage when blowing lead matte without siliceous ore were a very long converting time in the lead converter (2 hr. for partly blowing a charge of about 10 tons of lead matte) and considerable additional time in blowing the heavy, iron, residual material with copper matte

in the copper converter after this material had lost a good deal of its heat during the process of transfer from the lead to the copper converter.

3. Much of the slowness in blowing in the lead converter was due to the difficulty in keeping the tuyeres open; much punching was required and even then there was a marked tendency for tuyeres to blind.

4. Because of the large amount of magnetite formed in the lead converter, transferred to the copper converter and never completely slagged with siliceous ore in the latter, considerable difficulty was encountered in granulating the copper-converter slag, which was the practice for many years. This slag granulation was generally accompanied by violent explosions of magnetite, which often caused considerable damage to the granulator. The magnetite made slow granulation necessary and this in turn caused the formation of much shell in the granulator pot and launders, which was costly to handle and crush.

5. The presence of considerable amounts of magnetite in the converter slag made it impractical to treat this slag hot in the reverberatory furnace, as the magnetite would segregate in the bottom of the furnace and cause the usual troubles, making it impossible to get the full fluxing value from the iron in the converter slag. Because of this trouble in the reverberatory furnace, the converter slag was granulated and used at either the lead or copper plants as fluxing conditions demanded. When treated at the copper plant, the granulated converter slag was mixed as an integral part of the roaster charge and was fed to the reverberatory furnace as calcine. This method of handling, while costly, seemed to avoid much of the trouble from magnetite in the furnace that occurred when the converter slag was returned hot, and made practically all of the iron in that slag available to flux the siliceous ores in the roaster charge.

6. The high-iron residual material transferred from the lead to the copper converter not only contained much magnetite but also much copper. In the slagging of this residual material in the copper converter it seemed impossible to make a converter slag properly clean in copper and free of magnetite. As there were long periods at a time when the granulated converter slag was used at the lead plant, the presence of too much copper in this slag was particularly undesirable. This was really the primary cause for investigations, which in the last four or five years of this copper-plant operation completely changed the converting practice.

Siliceous Ore in Lead Converting

In 1927 it was decided to change the lead-converting practice by adding some siliceous ore to that matte in the same manner as in the converting of copper matte and, after skimming off the slag, blow to white metal in the lead converter; later combining this white metal with that made from copper matte to finish to blister copper. The average of both the lead-converter and copper-converter slag contained about

17 per cent SiO_2 during this period. At the same time, in order properly to protect the woolen bags in the baghouse from any acid in the lead-converter gases, the Sprague process, in which dehydrated lime is added to these gases just before they enter the baghouse, was used. In the former process, where no siliceous ore was used, the fact that the zinc in the lead matte was volatilized as ZnO instead of being slagged with silica provided the necessary neutralizing agent for free acid in the baghouse bags, and always provided very satisfactory protection. With the use of siliceous ores, however, much of the zinc went into the converter slag, leaving an insufficient amount volatilized to protect the bags and making the use of lime necessary. The use of siliceous ores in the blowing of lead matte was beneficial in several ways; particularly in decreasing the time of blowing, materially reducing the amount of magnetite and thereby copper in the converter slag, and reducing the proportion of shell to granulated converter slag. The copper assay of the converter slag treated at the lead plant was decreased by 3.5 per cent in early 1927 by the use of some siliceous ores in the lead converter. This fact alone fully justified the change, as much copper that formerly had been recirculated through the lead plant at high cost now found its way directly to blister copper. During this period (March, 1927, through June, 1928) the converter gases were handled the same as before; the lead gases were treated in the baghouse and the copper gases in the Cottrell treater.

Lime as Neutralizing Agent in Copper Converting

After observing for several months the satisfactory handling of the acid in the lead-converter gases by means of the Sprague process at the baghouse, experimental work was done in a small baghouse to determine with what success the copper-converter gases might be neutralized by the use of lime. The reasons for this investigation were the unsatisfactory results being obtained at the converter Cottrell treater where recoveries were none too satisfactory, and cost of upkeep, owing to excessive corrosion as a direct result of necessary humidification of gases, was extremely high. The results of these tests indicated that it was quite feasible to properly neutralize our copper-converter gases for baghouse treatment without the use of excessive amounts of lime. The same baghouse that had been used for years for lead-converter gases only was equipped with mechanical bag shakers, as well as with a fan of larger capacity, and was used to treat all the gases from both lead and copper converters, starting in June, 1928. The use of the Cottrell treater was entirely discontinued. The combined converter gases were now carried through a much longer flue system than the old one had been. This increased fan capacity, improved bag shaking and additional length of flue, in connection with the use of the old Cottrell humidification chamber where excessively hot

converter gases could be cooled with water sprays, made it possible to treat much larger gas volumes and dust load in this baghouse without any material delay due to high temperature. Also, lead and copper matte could now be blown together in any proportion in the same converter, which increased the time efficiency of the whole converting operation as a result of fewer delays in waiting for one particular kind of matte. With this type of converting and all the gases treated in the baghouse, about 100 lb. of lime was used per hour when blowing three of the 10-ton converters on lead and copper matte. As this lime costs about $0.85 per 100 lb. delivered at this plant, and is mechanically fed into the converter flue almost without expense, the cost for such neutralization is but little over $0.06 per ton of matte, with small converters treating 4½ tons of matte per converter hour. While the amount of lime used does not by any means neutralize all of the acid in the converter gases, it does neutralize any acid that collects with the fume and dust in the bags, and thereby satisfactorily protects the bags.

Increase in Amount of Siliceous Ore

The last change made in the converting practice was early in 1930, when a decided increase was made in the amount of siliceous ores used at the converters and a consequent increase in the silica content of the converter slag produced. This was done when converting a mixture of 50 to 65 per cent copper matte and 35 to 50 per cent lead matte, the former containing about 30 per cent Cu and the latter about 14 per cent Pb and 10 per cent Cu. The converter slag produced in the last six months of 1929 contained about 20 per cent SiO_2 whereas in 1930–1931 it contained about 28 per cent SiO_2. When making the former, lower silica slag, these converters treated about 105 tons of copper and lead matte per converter day and about 19 tons of siliceous ore. When making a 28 per cent SiO_2 converter slag, the same amount of matte was blown per converter day, but the amount of siliceous ore treated was increased from 19 to 25 tons per converter day. These data show that, with an approximately constant unit cost, the additional 6 tons of siliceous ores treated per converter day took practically no *additional* converting cost. The 20 per cent SiO_2 slag contained a little less than 3 per cent Cu and about 3 per cent Pb, whereas the 28 per cent silica slag was somewhat lower in copper content and somewhat higher in lead content; the totals for the two metals being very nearly the same in both cases. The higher lead content was due largely to the fact that the copper matte in the latter period contained more lead than in the former period, rather than to the change in amount of siliceous ores treated at the converters. Under similar operating conditions, we believe that the lower copper assay of the higher silica converter slag more than compensates for the

additional amount of slag produced, thus somewhat decreasing the total amount of copper recirculated in converter slag.

The reason for this change in practice in 1930 was a change in our lead-smelting and copper-smelting operations which required the treatment of most of the converter slag at the copper plant rather than at the lead plant. Developments in practice at other copper smelters had clearly shown that a sufficiently high-silica, magnetite-free, converter slag could be successfully and economically treated hot in a reverberatory furnace treating a regular calcined charge, and full fluxing value of the iron in the hot converter slag could be obtained. Our experience fully justified this view, as we were able to treat as much or slightly more calcined charge per furnace day at the reverberatory furnace in addition to the hot converter slag as we had done before returning this hot slag to the furnace. Also, full fluxing value of the iron in the converter slag was obtained and reverberatory slags were as clean as ever. Thus by the use of more siliceous ore in the converters at practically no additional cost the reverberatory furnace capacity for new material was increased by 15 to 20 per cent and the expense of granulating, handling and "roasting" about 75 tons per day of converter slag was avoided.

Table 1 shows converter data by months for the eight months of operation in 1931, during which period a high-silica converter slag was produced.

Handling Converter Gases

As briefly described above, one of the important developments at this converting plant was the method of handling the gases from the converters. When operating under the original plan of handling only lead-converter gases at the baghouse, a No. 20 Sirocco fan driven by a 75-hp. motor was used to deliver a volume of about 80,000 cu. ft. per minute against a 2½-in. static head to the 960 bags. These bags are 18 in. in diameter and 30 ft. long, but when hung by the old method (gathered near the top and tied to a metal "stirrup" hanger), the effective filtering length was only equivalent to 27 ft. The effective filtering-bag area for the baghouse was 122,150 sq. ft. When handling 80,000 cu. ft. of gas per min. there was 0.65 cu. ft. per min. filtered per square foot of bag area. The fume recovered per square foot of effective bag area per day averaged about 0.35 lb. under these conditions, or nearly 45 lb. per bag day. During this period the gas pressure in the bags was kept under 2 in. of water by shaking three or four times each shift. This shaking was accomplished by the reverse-current method, in which the gas flow was shut off from one pit at a time and a suction fan used to collapse the bags; the alternating inflation and deflation loosening the fume in the bags and permitting it to drop into collection pits below the thimble-floor. The life of woolen bags under this kind of operation averaged 2½ years,

TABLE 1.—*High-silica Converter Slag Period*, 1931

Month	Analysis, Per Cent									Converter Days	Average Converter Days per Day	Tons Blown per Converter Day							Tons Slag per Converter Day
	Converter Slag				Lead Matte			Copper Matte											
	Cu	Pb	SiO_2	Fe	Cu	Pb	Zn	Cu	Pb			Cu Matte	Pb Matte	Cold Pb Matte	Custom Matte	Dope	SiO_2 Ore	Total	
January	1.6	4.2	30.8	44.2	9.7	15.4	2.6	27.5	8.2	41.13	1.33	73.6	26.8	2.63	3.00	13.2	24.3	143.53	76.00
February	1.6	4.1	28.3	45.9	9.3	12.7	2.3	28.0	8.1	40.84	1.46	68.7	30.0	0.68	5.12	13.1	23.4	141.00	74.10
March	1.6	4.6	28.6	47.0	9.0	12.1	1.7	27.5	7.8	47.57	1.53	72.8	26.0	0.76	1.47	16.8	25.0	142.83	78.00
April	1.5	4.8	29.2	45.7	9.5	11.5	1.7	28.2	8.6	41.87	1.39	69.5	30.3	0.79	1.43	17.0	24.7	143.72	78.00
May	1.7	3.7	27.0	48.6	9.7	15.4	1.7	28.9	7.3	36.40	1.17	70.3	27.4	0.77	2.80	13.1	23.1	137.47	72.30
June	1.8	4.2	24.3	47.0	7.4	17.3	2.1	27.3	7.9	34.27	1.14	74.4	25.8	1.22	8.93	6.9	24.3	141.55	73.80
November	1.7	5.2	27.7	46.6	7.2	21.7	1.9	26.4	7.3	38.23	1.28	80.9	23.4	0.52	5.13	11.9	25.5	147.35	77.20
December	1.4	4.2	28.5	45.8	6.9	19.6	1.9	22.8	7.6	42.84	1.38	82.1	17.7	0.60	4.65	15.4	26.7	147.15	77.80
Average	1.6	4.9	28.0	46.3	8.6	15.7	2.0	27.1	7.8	40.39	1.33	74.0	25.9	1.00	4.08	13.4	24.7	143.08	76.00

which was satisfactory under the conditions. The cost of bags per ton of fume filtered and recovered was about $0.70, based on the average cost of the woolen bags at that time—about $12 each.

After the converting practice was changed in 1928 to make it possible to blow lead and copper matte together, the same baghouse had to handle greatly increased volumes of converter gas, as well as greater dust loads. This was accomplished by four means: (1) The flue between converters and baghouse was lengthened from about 300 ft. to over 800 ft., including a brick humidification chamber 645 ft. ahead of the baghouse, in which high-pressure water sprays could be used to cool the gases when temperatures were too high. (2) The old fan was replaced with a new design No. 17 Sirocco fan capable of delivering 200,000 cu. ft. against a static head of 4 in. of water. This fan is driven by a 175-hp. motor. (3) The baghouse was equipped with mechanical shakers, replacing the old reverse-current method of shaking. (4) Top "thimbles" were installed on all bags, replacing the old method of bag suspension with one that permits each bag to hang from top to bottom in cylindrical form and does away with the top folds that existed with the former method of suspension. The installation of these top "thimbles" increased the available filtering area of these bags by over 7 per cent, or from 122,150 sq. ft. to 131,400 sq. ft. These top thimbles were installed mainly to lengthen bag life with the operation of mechanical shakers, as this type of shaking was found to be very hard on the bags when they were hung by being tied over stirrup hangers, the bag material wearing with undue rapidity on the overlapping folds near the top. The use of these thimbles had the desired effect of materially increasing bag life as well as available filtering area.

Since these changes were made this baghouse will handle approximately 145,000 cu. ft. of gas with an average pressure in the bags of 2½ in. of water. This is accomplished by mechanically shaking the bags five or six times each shift. With the present available filtering area of bags at 131,400 sq. ft., the gas filtered per square foot of effective bag area per minute is slightly over 1.1 cu. ft., or about 70 per cent more than it was before the changes were made. The fume recovered per square foot of effective bag area per day, with normal operations, now averages nearly 0.4 lb. (or about 50 lb. per bag day).

Considered in all phases, these developments in operating practice and gas-handling methods at the Tooele converting plant have resulted in considerable increase in efficiency and saving in operating cost.

The Messina Stationary Basic Copper Converter

By R. G. Knickerbocker,* Rolla, Mo.

(New York Meeting, February, 1932)

The copper smelter and refinery of The Messina (Transvaal) Development Co. Ltd., at Messina, South Africa, was erected in 1920 and 1921, but initial operations were deferred until late in 1922 on account of the collapse of the copper market in the interim.

Because the anticipated production was to be only 20 tons of copper per day from a high-grade matte containing 60 per cent copper, and because the ore did not contain gold or silver in quantities sufficient to pay for separation from the copper bullion, it was decided to design the plant along lines quite different from the modern smelteries treating large quantities of sulfide ores carrying appreciable amounts of gold and silver. The plant was built, therefore, for the production of copper by the Nicholls-James process. By this method two-thirds of the copper matte produced was roasted in a Merton calciner and charged into a bath of the remaining one-third portion of unroasted matte in a reaction furnace.

The original design was not only novel in providing for the Nicholls-James process, but also in comparison with new plants built elsewhere, because of factors inherent in the locality. Sources of supply for machinery were very remote; it takes three months after the placing of an order to accomplish delivery of machinery into Messina from the United States. Native labor was available at 30 to 40¢ per 8-hr. shift. These factors permitted plant design which provided no overhead cranes, no matte ladles, no slag pots or other usual mechanical means of transportation between departments. The design resulted in a compact simple plant with furnaces arranged close together and terraced so as to provide for gravity transfer of molten material from one furnace to another and for manual disposal of products. Fig. 1 shows the relative location of furnaces. The three furnaces are parallel on successively decreasing elevations. A superimposed feed floor was provided above the furnaces. All three furnaces were provided with coal-dust burners served from a Bonnot system of coal-dust transportation, and the waste gases were discharged into waste-heat boilers.

A short trial of the Nicholls-James process developed several facts which brought about a decision to abandon the process. Owing to the

* Engineer in charge of erection at Messina, 1920–1921; Chief Metallurgical Engineer in charge of operation, 1923–1926.

percentage of iron in the matte the plant provided insufficient calcining capacity and insufficient smelting capacity in the reaction furnace, so that no more than 100 tons of copper per month was produced instead of the amount for which the plant had been designed.

In first attempting to depart from the Nicholls-James process, we tried to blow the matte to white metal and then to blister copper by the use of protected blowing pipes inserted through the side door of No. 2 furnace, and into the bath. Two types of protected blowpipes were used. In the first trial the pipes were covered with refractory sleeves made of fireclay and also of magnesite. In the second attempt, oil-cooled sleeves were installed over the blowpipes. In both of these experiments the splash of slag and metal which accumulated on the pipes was so serious that it was decided to attack the problem in a different manner.

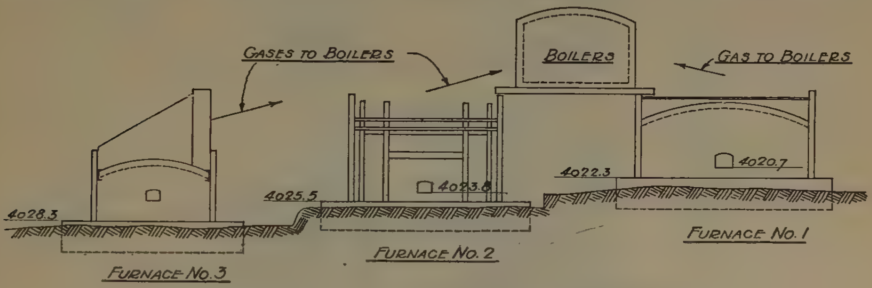


FIG. 1.—RELATIVE ARRANGEMENT OF FURNACES.
The numbers 4028, etc., refer to an arbitrary datum plane.

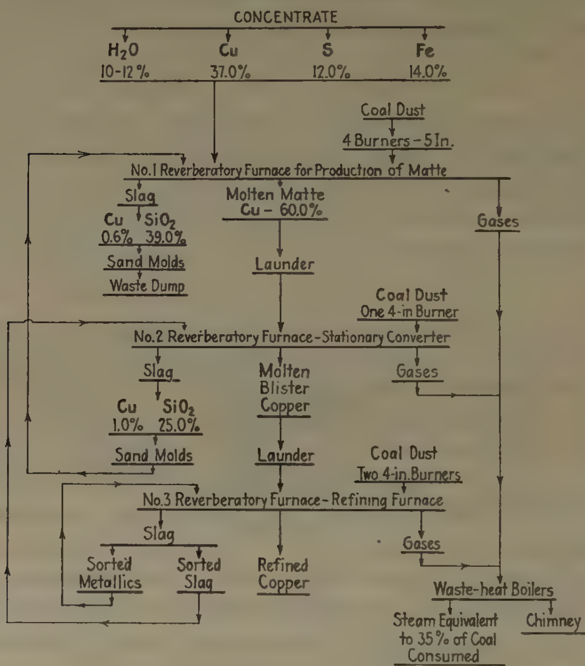
Naturally, consideration was given to the proposal to adopt standard converting methods, but after contemplating the purchase of a rotating or tilting converter shell and of accessory cranes, ladles, etc., it was decided that time and money would be saved by developing a stationary converter from one of the existing furnaces. No. 2 reverberatory, the reaction furnace of the Nicholls-James process, was selected for remodeling into a stationary converter. It is the primary purpose of this paper to describe the design, operation and economy of the whole smelting and refining practice that resulted.

Theory and Principles

The feed to the smelter consisted of a supply of flotation and gravity concentrates containing: 10 to 12 per cent H_2O, 37 per cent Cu, 12 per cent S and 14 per cent Fe. Obviously a high-grade matte would result and, as stated above, there was insufficient gold and silver to make electrolytic separation worth while. The problem, therefore, was that of converting small volumes of matte which would be of high grade, both conditions being disadvantageous for maintaining sufficient temperature without the use of extraneous fuel. An incidental problem was to transfer the small volumes of matte and blister copper with a minimum

amount of solidification during the transfer. The problem was solved, as described below, using only the previously existing furnaces, and resulted in smelting and refining practice according to the accompanying flow sheet.

Flow Sheet at Messina



Design and Construction

Obviously, the major problem in remodeling No. 2 furnace into a stationary converter was that of providing air-blast tuyeres which could be easily plugged when it would be necessary to interrupt the flow of air while the tuyere apertures were still submerged under a bath of 16 to 18 in. of molten metal.

No. 2 furnace as originally built had a 9-in. magnesite inverted arch hearth with 13-ft. radius superimposed upon a 12-in. silica brick "first" bottom, which was laid on firebrick and fireclay. The refractories were underlaid by a concrete slab 18 in. thick supported by a bed of coarse broken slag 2 to 3 ft. deep, laid down on bedrock. The furnace hearth was 11 ft. wide by 15 ft. long. The roof arch, at the burner end, on the longitudinal center line was 6 ft. above the magnesite bottom and sloped downwards toward the gas uptake, at which point it was 3 ft. above the bottom. The roof arch and the side walls above the metal line were of silica brick, while below that point the side walls consisted of a silica-brick casing with an inside lining of 9-in. magnesite brick.

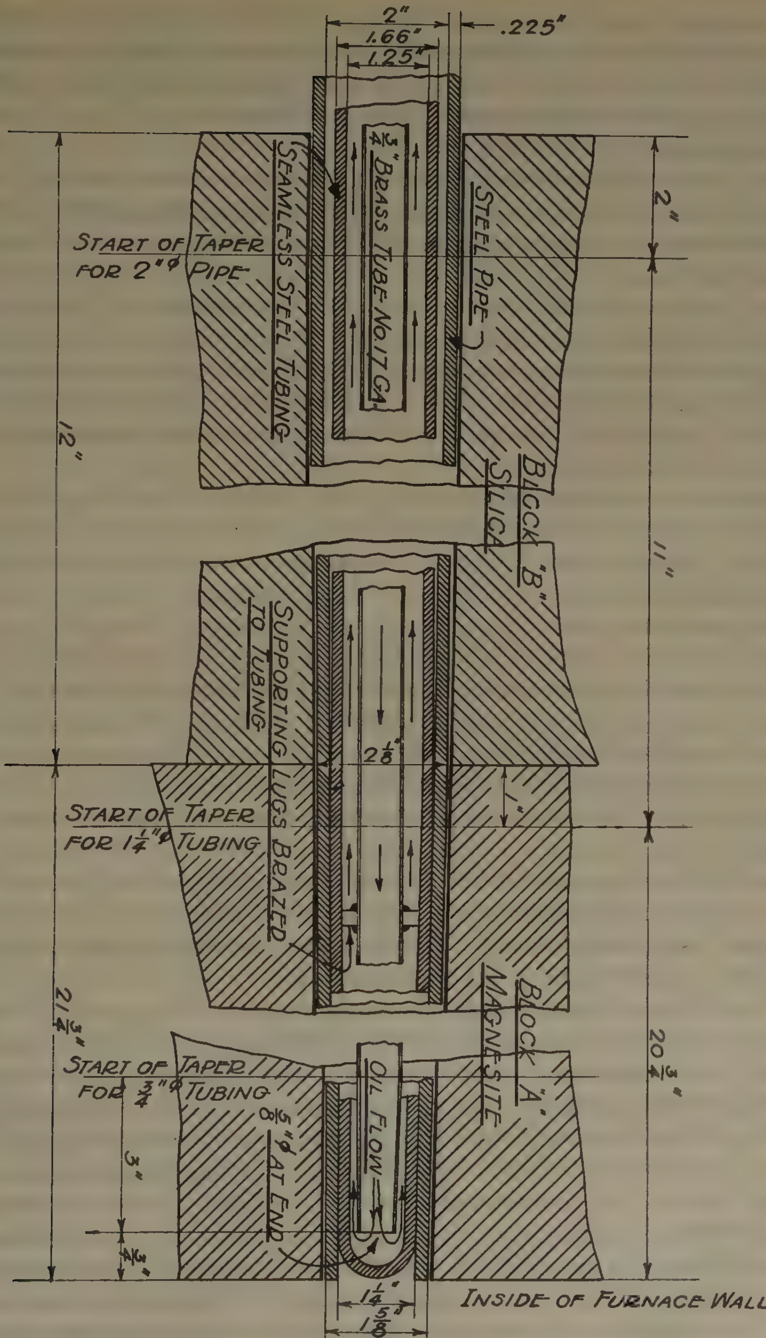


Fig. 2.—Details of oil-cooled tuyere plug.

The buckstays were of steel 8-in. H-beams and were tied across the top of the furnace by steel rods, while the bottom ends were held by reinforced shoulders of the concrete slab. All the magnesite and silica brick used in construction were imported from the United States. No difficulty was ever experienced with any bottoms.

The remodeling consisted of the removal of the burner wall of No. 2 furnace from the metal line down to the top of the silica first bottom and the substitution of the tuyere wall backed up by a steel tuyere plate. While 14 tuyeres were first installed it was soon found that 7 gave better action and stronger wall construction. The tuyeres were 1¼ in. at the discharge aperture of the new tuyere pipe. The tuyeres were served from a 7-in. dia. air main, connection being made to each tuyere pipe through a standard Dibblie ball valve. The valves were mounted on I-beams, but were free to move to accommodate themselves to the expansion and contraction of the furnace.

The tuyere problem was solved by the design and development of oil-cooled plugs to be inserted into the tuyeres before the air pressure is reduced. This is shown in Fig. 2. Steel pipes, swaged down on a mandrel to the tapered shape, were built into the refractory wall. The oil-cooled removable plugs consisted of two concentric tubes; the inner one being a ¾-in. brass tube for the incoming oil and the outer one a length of seamless steel tubing, also swaged down on a mandrel to the required taper and having the end closed by welding. Welding practice was not very good at Messina and one or two defective plugs resulted. At first ordinary lap-weld steel pipe was used for the outer wall of the oil-cooled plug, but it was found that these lap joints would open on account of the difference in temperature inside and outside of the wall at the point of contact with the molten bath. The resulting leak of oil into the bath made an oil burner out of the oil-cooled plug and, as the ruptured plug always stuck, a few runouts of metal occurred and the change to seamless tubing was therefore made.

The oil-cooled tuyere plug necessitated accessory apparatus: there being provided an oil-circulating system consisting of an oil pump, water-cooled "heat exchanger" and pump, strainers, sight flow feeders, and inlet and outlet piping to and from the plugs. The plugs were connected by 6 ft. of flexible metallic tubing to the swinging inlet and outlet piping 8 ft. back of the ball valves. The lengths of piping dropping from the charge floor to the converter floor were connected by ball and socket joints. The plugs hung on a rack when not in use. It was necessary to supply 10 to 12 gal. of oil per minute at 8 lb. pressure per square inch for circulation through each plug when in use. With this flow one could grip for a short time in the bare hand the oil-cooled plug just outside the ball valve when the plug was in service, even at the time of maximum heat. The oil originally used was ordinary fuel oil, which had a low

flash point and carbonized on the walls of the plugs, forming a nonconductive coating which allowed overheating of the walls and caused their ultimate failure and consequent leakage of oil into the furnace. This oil was replaced by a thin oil known as Velocite C, with a flash point of over 300° F. and a cooling capacity twice that of the fuel oil originally used. During the last two years that the author was in charge of operations, no trouble was experienced from leaky plugs and no runouts occurred. We did try using steam for the cooling medium but became dubious about this method on account of the possibility of scale forming in the inner space between the inlet and outlet walls at the "hot points."

The tuyere pipes were horizontally installed and the plane of the row of tuyeres was 12 in. above the magnesite bottom at the center, but as the bottom was an inverted arch the outside tuyeres were only 7 in. above the magnesite bottom.

The skimming door was in the end of the furnace opposite the tuyere wall. The tapping hole was also at the skimming end, 15 ft. from the tuyeres. In plan, the hearth of the furnace was rectangular. A charge-hole was provided for quartz flux at a point 3 ft. in front of the tuyere wall, this material being fed down through a chute from the charging floor.

Operating Practice

The operation of a stationary converter made it a matter of prime importance to avoid the flooding of tuyeres with molten bath. This contingency made it necessary to install an electrical signaling system between the power house and smelter to insure prompt and accurate air-blast control. The siren system employed permitted the repeating of signals back to the originator.

The cycle of operations in the stationary converter is as follows: The furnace being empty and heated to approximately 1000° C., the tap-hole plugged, tuyeres reamed, oil circulating properly and oil-cooled plugs inserted up to the marks, the power house is signaled to start the converter compressor and to stand by ready to give full capacity of air. Keeping the air pressure at 3 lb. per sq. in. on the gage at the tuyere windbox, the molten matte is transferred into the converter through a tamped sand launder from the matte furnace. As the matte rises above the ends of the plugs in the tuyeres, the air blast continues for a minute or so, but gradually ceases as the tuyere apertures are curtained off by the formation of a thin shell of frozen matte. When sufficient matte is transferred the tap-hole is plugged with clay and the power house is signaled that all is ready for opening the tuyeres. As soon as receipt of the signal is acknowledged, the main air valve is fully opened, all oil-cooled plugs are pulled out and hung on the rack out of the way of the tuyere punchers. The tuyeres are then all punched until they are blowing freely. The maximum bath level is 18 in. over the level of the tuyere

apertures and the corresponding pressure on the wind box is approximately 7–8 lb. per sq. in. About 15 min. after the blowing begins the coal burner is shut off, as the furnace reaction heat is sufficient to maintain the temperature after the reactions have been well started. Quartz flux is added through the port 3 ft. in front of the tuyere wall. When the iron of the bath has been oxidized and slagged, the tuyeres are reamed, the oil-cooled plugs are inserted up to the predetermined marks (the positions of which vary according to the wear of the furnace lining at the tuyere line), and the air pressure is gradually brought down to 3 lb. per sq. in. As the plugs are inserted into the tuyeres, the air blows through between the plug and the walls of the tuyere pipe, but a minute or so after the pressure is cut to 3 lb. the ends of the plugs are closed over with a thin scab of frozen metal, thereby sealing the tuyeres. The 3-lb. air pressure is maintained during this nonblowing period. Most of the converter slag is then run off by cutting down the clay breast at the skimming door and the remainder is rabbled off. The slag is run into a sand bed away from the front of the furnace. After skimming, another charge of matte is transferred from the matte furnace and the cycle is repeated until the converter is full of white metal. The blow from white metal to blister copper takes more punching to keep the tuyeres open and is deficient in heat from the reaction, so that during the last hour of this blow the coal burner is kept burning at full capacity. The finished blister copper is transferred from the converting furnace into the refining furnace without intermediate casting or solidification, the transfer being made through a tamped sand launder.

Before beginning another cycle of operations (and, in fact, before plugging each time) it is necessary to ream out the tuyeres to remove an accumulation of matte and copper which creeps back into the tuyeres and is not completely removed by the regular punching operations. This operation is accomplished with a long handled tool-steel reamer. Each day the positions of the marks on the tuyere plugs are revised to conform with the progressive wear of the furnace wall and the plugs are inserted into the tuyeres up to these marks. A special jackscrew is at hand for pulling plugs which stick badly. A quarter turn on an 18-in. handle at right angles to the pipe ordinarily is sufficient to loosen the plug. The plug ends are coated with graphite, bone ash or lime before each insertion.

When refractories are imported from the United States, the repairs are not heavy. The original 12-in. silica roof arch lasted only a short time and was replaced by a 9-in. arch of magnesite brick. This humped and buckled badly but held together because of the slag and metal absorbed. This roof is patched when necessary by laying in new bricks over a steel plate supported on chains and may be accomplished between charges, so that little tonnage is lost on account of arch repairs.

The tuyere wall averaged 2500 tons of blister copper before being renewed. The renewal period required five days. No other repairs were needed on side walls or elsewhere during the author's supervision.

This is the only installation of which the writer has knowledge of converter gases passing through waste-heat boilers. These gases combine with those of the matte and the refining furnace and go through the same boilers. There has been no boiler trouble. Approximately 35 per cent of the fuel burned in the furnaces is recovered in the form of waste-heat steam, in spite of the small size of the furnaces and consequently larger proportionate radiation losses.

There are no detrimental impurities in Messina blister copper which prevent it from being fire-refined to a quality that meets the A. S. T. M. specifications, but production of this quality was not realized except for occasional, long-time charges, because during the Nicholls-James operations calcines and raw matte were smelted in the No. 3 refining furnace and the bottom absorption of sulfur and iron prevented cleaning of the bottom and made it impossible to obtain a clean low-set button. This magnesite bottom was saturated with matte and blister down to the last inch of depth. The author has always felt that this should have been replaced by a silica bottom, and that two or three 1¼-in. tuyeres with oil-cooled plugs, if installed in the end wall 12 in. below the metal line, would have made possible a saving on blowing time and pipe and the production of higher grade copper. The ordinary blowing of this molten blister bath through side doors in the refining furnace by the use of ¾-in. bare iron pipes inserted into the bath takes three times as long as a remelted blister bath.

The operating force is composed of local native labor and white foremen. None of them ever saw a smelter or refinery before they worked at Messina.

TABLE 1.—*Treatment and Production at Messina*

Dry, Daily Charge to Matte Furnace, Tons	Total Coal Daily, Tons	Copper Concentrates, Per Cent	Wet Charge per Ton Dry Coal, Tons	Ingot Copper Produced, Tons per Month	Recovery, Per Cent
80.0	22.0	37.0	4.0	625.0	95 plus

COSTS

As already stated, native labor is available at 30 to 40¢ per 8-hr. shift. White foremen are paid $150 per month. This is taking the pound sterling at par value. Coal costs $6 per long ton delivered at the railroad siding. Total smelter and refinery costs for the last half of 1925 were approximately $30 per ton of ingot copper produced. This

cost covers the operations from unloading concentrates to and including loading of ingot copper into railroad cars. It does not include taxes or depreciation.

Conclusions

1. The revised Messina practice demonstrates that high-grade copper matte can be economically converted to blister copper in a stationary converter with the intermittent use of additional heat and without the use of machinery in the usual standard converting practice.
2. Refined copper meeting A. S. T. M. specifications can be produced from molten Messina blister copper without allowing it to solidify before charging to the refining furnace.
3. The Messina practice is good for small or medium sized copper producers where the available labor is very cheap and not highly skilful.

Mr. A. B. Emery, General Manager of the Messina Co., deserves considerable credit for this economical metallurgical plant and for his ability to inspire his men to unusual application. In spite of high mining costs, high fuel costs, with machinery and supply replacements being imported from the United States, and with a production of only 20 tons of copper per day, he has, for the past eight years, competed with the costs of some of the larger producers in the western United States.

DISCUSSION

(*Francis R. Pyne presiding*)

C. R. Kuzell, Clarkdale, Ariz. (written discussion).—Mr. Knickerbocker's paper is interesting not only because it records the successful solution of the problem confronting a small copper producer and because the result was obtained at a favorable unit cost, but also by reason of the disclosure of several interesting features in smelting practice.

Almost every smelterman has contemplated the problem of applying tuyeres to a nontilting or rotary furnace. The success attained at Messina will undoubtedly stimulate engineers to reopen projects temporarily laid aside because such projects depended upon the application of tuyeres for the injection of air or gases into the bath of a stationary furnace. The oil-cooled tuyere plug evidently did not cause any disasters as the use of water might have done. Mr. Knickerbocker said that the failure of the oil-cooled tuyere plug resulted in a flare of oil flame in the furnace. This suggests the substitution of oil instead of water in fluid-cooled furnace parts, particularly where there is danger attendant upon the failure of a water-cooled device in contact with or adjacent to a molten metal bath.

The fact that the converter gases pass through waste-heat boilers and that no trouble was encountered is also good news to the metallurgist who has wanted to try such a scheme. Unfortunately, at Messina the converter gases apparently discharged into a flue common to the other furnaces served by the waste-heat boilers, therefore no data are given on the yield of waste-heat steam from the converting process itself.

The most recent practice in the smelting of concentrates indicates a strong trend toward feeding the wet concentrates directly to the reverberatory furnace for the production of matte. The revised Messina flow sheet (p. 142) indicates that concentrates containing from 10 to 12 per cent of moisture were fed directly to the reverberatory furnace.

Smelting Copper Concentrates in a Converter

By George E. Beavers,* Copperhill, Tenn.

(New York Meeting, February, 1934)

Under the title used for this paper, the converter practice of the Tennessee Copper Co. is described in the Transactions of the Institute.[1] That article contains the following statement: "The limit in capacity of the Peirce-Smith converter in terms of concentrate has not been reached but approximately as high as one ton of concentrate has been handled for each ton of liquid matte charged and good working conditions have been maintained."

It is not the intention of the writer to review the conditions leading up to the introduction of a Peirce-Smith converter at Copperhill, but, instead, to offer later data to show how satisfactory the installation has been.

Converter Campaign, June 15, 1930 to August 23, 1931

Blowing time (converter days)	349.90
Number of blows	427.00
Tons matte charged	38,335.00
Copper assay of matte, per cent	13.67
Tons blister copper made	9,246.00
Time to blow one ton of copper, minutes	54.50
Air used per minute, cu. ft.	12,500.00
Blast pressure, lb.	12.80
Air per ton copper, cu. ft.	681,330.00
Air per ton iron, cu. ft.	199,040.00
Tons flux used	11,845.00
Tons flux per ton matte	0.31
Tons flux per ton copper	1.28
Tons flux per ton iron	0.37
Tons concentrate treated	28,960.00
Copper assay of concentrate, per cent	17.13
Tons concentrate per ton matte	0.76
Tons material charged per converter day	235.70

* In charge of research, Tennessee Copper Co.

[1] F. J. Longworth: Smelting Copper Concentrates in a Converter. *Trans.* A.I.M.E. (1925) **71**, 969.

Converter Slag Assay, Per Cent

Copper	1.34	Alumina	0.70
Iron	55.30	Silica	19.90
Sulfur	2.00	Calcium oxide	0.50

Slag Disposal per Converter Day

Tons liquid slag poured into blast-furnace settler	119.90
Tons converter slag skulls smelted in blast furnace	46.10
Total tons converter slag poured	166.00

Slag Losses

Total settler slag averaged about seven parts of furnace slag at 0.27 per cent copper and three parts of converter slag, the combined slag assaying 0.35 per cent copper. The copper loss in converter slag was approximately 22.0 lb. per ton iron in charge, or 8.0 lb. copper per ton of concentrate smelted, with concentrate of this grade, the net copper recovery after allowing for dust losses was 97.0 per cent.

Ratio of Concentrate to Matte

This campaign represents the highest concentrate to matte ratio yet maintained; that is, considering the length of the campaign. Below is the ratio of concentrate to matte by years:

Year	Ratio	Year	Ratio	Year	Ratio
1925	0.658	1928	0.585	1930	0.795
1926	0.711	1929	0.633	1931	0.695
1927	0.719				

Sulfuric Acid from Converter Gases

By George E. Beavers,* Copperhill, Tenn.

It is well to understand at once that converter gas is always mixed with copper blast-furnace gas at Copperhill before entering the Glover tower, and, at times, roaster gas is also added along with the converter and blast-furnace gas.

Fig. 1 is introduced to show the variations in the sulfur dioxide content of the gas and the ability of the acid-plant operator to control

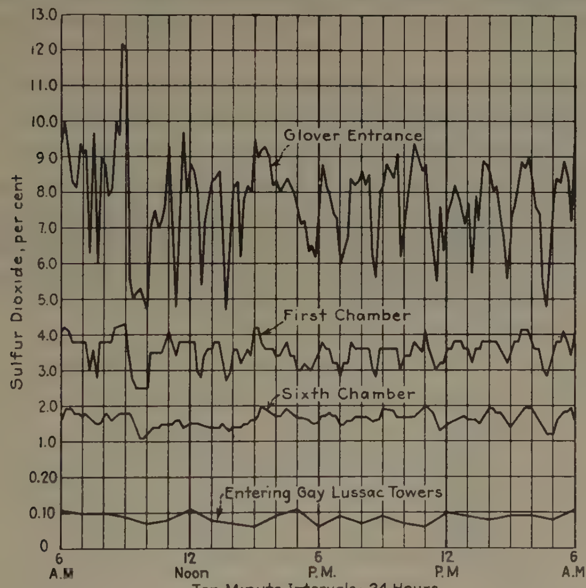


Fig. 1.—Fluctuations of sulfur dioxide content of converter gas.

the variations, resulting in a very even grade at the Gay-Lussac tower entrance. The sulfur dioxide content of the gas entering the Gay-Lussac tower must be low and regular to assure low niter consumption.

The blast-furnace gas alone is rather uniform as regards sulfur dioxide content and the variations, as shown in Fig. 1, are chiefly caused by converter changes—out for charging, slag pouring, copper pouring, and, of course, the "finish." The combined gas during the converter finish is, at times, almost free of oxygen and the sulfur dioxide content reaches 13.0 per cent. At such times air is added at the Glover tower.

* In charge of research, Tennessee Copper Co.

While the gas from the blast furnace contains some carbon dioxide, the converter gas is very pure, and the mixture, therefore, is comparatively easy to handle. The recovery of sulfur into acid from the converter gas is around 95.0 per cent and the recovery of the sulfur eliminated at the converter into acid is 97.0 per cent or better. When producing around 850 tons of 60° acid daily in No. 1 acid plant, the converter accounts for 80 to 120 tons.

The converter gas is so clean and pure that it tempers the blast-furnace gas to a point where the combined gas has qualities as desirable as the blast-furnace gas alone, in spite of the variations in the converter gas.

The niter is introduced from ammonia oxidation units and from stocks of nitrous vitriol. Whenever possible the variations in the gas are handled by changes in the amount of nitrous vitriol charged. However, at times, such as the end of the converter finish, the niter from the ammonia oxidation units must be cut off and great cuts in the nitrous vitriol must follow.

The chamber operator, by means of remote-controlled charts and signals, has intimate knowledge of converter changes at all times.

Owing to the carbon dioxide content of the gas, the Reich test is applied in determining the sulfur dioxide content of the gas. The gas is tested at ten-minute intervals at the Glover entrance, No. 1 chamber, No. 6 chamber and at Gay-Lussac tower entrance. This is demonstrated in Fig. 1. This information aids in the control of the process.

Development of Monolithic Tamped Periclase Converter Linings at United Verde Copper Company Smelter

By F. H. Parsons,* Clarkdale, Arizona

At the time that converting or bessemerizing of copper matte first began to be practiced by the smelters in the copper industry, converters were lined by tamping the flux, usually siliceous ore, into the steel shell. About 1910, practice in several plants had demonstrated the superiority of a basic lining of nonfluxing material and the addition of flux through the mouth of the converter. Adoption of the new method by nearly the entire industry followed. Basic linings are made of refractory "magnesite" brick manufactured, usually, from imported dead-burned magnesite. Either the linings are laid up in the shell dry or the bricks are bonded with some mortar dictated by the preference of the user.

Foreign, usually Austrian, magnesite was the main source of supply for these bricks, although there were domestic sources, but it was not until costs began to rise a few years after the first use of basic linings that these domestic sources were investigated to any great extent. Exigencies created by the World War necessitated the use of the domestic magnesite during this period, but the brick manufacturers reverted to the use of the foreign product after the war, because the impurities therein made a brick with a better fire bond than the purer domestic magnesite. Therefore the domestic product became available at a favorable price for any metallurgical process that permitted its use.

The United Verde converting practice using magnesite linings has resulted in a high consumption of magnesite because of the zinc content of the low-grade matte. It was realized that the cost of these linings represented a large portion of the cost of converting, and an investigation was started to find means by which these costs might be lowered.

In the neighboring state of California rather pure magnesite was being mined and treated for the market. The product, however, was purposely made to simulate the dead-burned grain magnesite product from foreign sources by the addition of fluxing impurities before burning. It was learned that this dead-burned California grain magnesite could be laid down in Arizona at a price one-third that of the same tonnage of "magnesite" brick manufactured from foreign magnesite. It was also learned that at least one California producer was ready to supply the market with periclase, which was a grain magnesia produced by burning the

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* Assistant Smelter Superintendent, United Verde Copper Co.

high-grade magnesite, without fluxing impurities, at a very high temperature. The periclase is a dense crystalline grain, more resistant to weathering, chemical action and erosion. It was available in grades containing from 88 to 92 per cent MgO.

Owing to patent litigation, the United Verde company had used acid linings during the years 1925 and 1926; had perfected mechanical means for tamping linings, and had trained operators and helpers to do the work.

With such a set of favorable conditions, the United Verde undertook the experiment of substituting monolithic tamped linings of grain magnesite or periclase to replace the linings of manufactured magnesite brick.

The first converter to be lined under the new method was tamped late in September, 1928. The mix for the lining was crushed magnesite brickbats, a quantity of which was on hand, and fireclay, but because of fluxing impurities, use of too much clay and too much water, this lining lasted only five days. This was followed by several shells lined with tamped linings of the dead-burned grain magnesite that simulated the Austrian product. The results were encouraging and the next step was the use of grain periclase. This purer, and otherwise more desirable, material was so superior that it was used in all subsequent trials. The work was conducted under direction of C. R. Kuzell and M. G. Fowler.[1]

Early in 1929 the lengths of the campaigns of tamped converters showed marked improvement, due in some measure to improved technique in mixing and tamping, and largely to recognition of the importance of grain sizing. At this time, therefore, more shells were put into this service and development progressed much more rapidly.

The importance of a proper binder was recognized early in the investigation, and among those tried were: (1) fireclay, (2) crude bauxite, (3) calcined bauxite, (4) sulfuric acid, (5) boric acid, (6) blackstrap molasses, (7) crude oil sludge. Also, combinations were tried, of 1 and 4; 1, 2, 3 and 4; 4 and 6; and 5 and 6. The practice resolved itself into the use of either of two preferred mixes, these being periclase, fireclay and sulfuric acid, or periclase, molasses and sulfuric acid.

At one stage of the experimental work an attempt was made to tamp the lining by hand without removing the converter cap. The tamping was exceedingly arduous and probably was not well done. Early in 1930 the standard procedure, to be described later, was adopted, and since that time all converting at United Verde smelter has been carried on in shells with tamped periclase linings, except that one shell lined with magnesite brick has been blown in regular turn so that comparison between performances of brick-lined and periclase-lined converters has always been possible.

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[1] Three patents have been applied for, one of which has been granted to Morris G. Fowler as U. S. Patent No. 1893023.

Theory and Principles

During preliminary work in the solution of the problem of tamping a monolithic periclase lining into a converter shell many difficulties were encountered. The first major question involved appeared to be the proper grain sizing of the mixture. Should there be a definite proportion of each grain size, or must the coarser particles and finer sizes bear a definite relationship? Many laboratory tests were made to determine this point. It was finally decided that a maximum of 22 per cent of minus 100 mesh and a minimum of 12 per cent was necessary to give the highest mechanical strength. It was then found that, all other things being equal, magnesite with the greater proportion of large grains gave best results. In the tamped linings best results have been obtained when the mix contained a large proportion of the coarse material, relatively few intermediate grains and about 14 per cent of minus 100-mesh material.

The question of what bonding material to use was a matter of constant experimentation. Fireclay gave the best tamping bond but had very low dry strength with the periclase and required a relatively high temperature to give it a strong fire bond. Molasses showed a greater dry strength than either fireclay or sulfate, and retained its bonding strength to a much higher temperature, the periclase remaining well bonded after it was heated above the temperature at which the molasses burned out. The reason for this is possibly the sintering of the finer periclase assisted by partial reduction of the iron by the molasses. At one stage of these tests bauxite was used as a binder with dead-burned magnesite but it did not work well because there was too much silica in both the bauxite and the magnesite fines to give proper composition for a spinel of high melting point. Boric acid was tried as a bond, but gave very unsatisfactory results, as is shown later in the section on costs. During the last year of this work only fireclay or molasses with sulfuric acid has been used as a binder. The use of sulfuric acid has been determined to be of such importance that the following paragraphs regarding it have been contributed by M. G. Fowler, of United Verde research staff.

The Importance of Acid

The periclase used at the United Verde for tamped basic linings contains over 2 per cent of lime. In order to tamp well, the mixture of periclase, tamping bond and water must be reasonably plastic. It has been found in clay technology that a relatively slight change in the hydrogen-ion concentration of the water-clay mixture, within a small range, has profound effects on the plasticity of the mixture. Strongly alkaline or strongly acid solutions cause complete deflocculation and

almost complete loss of plasticity. Maximum plasticity is generally attained close to the point of neutrality of the solution, and at this point the liquid content of the mixture can be varied over the widest range with retention of plasticity of the clay. These facts hold true for most plastic materials made up with water. The degree of flocculation, and thus the plasticity, depends upon the strength of the charges of the colloidal particles, and upon the hydrogen-ion concentration of the solution.

Since plasticity is essential to good tamping, it becomes necessary to eliminate, or to neutralize, any impurities that will dissolve in the water used and cause deflocculation of the colloidal materials used. As mentioned above, the chief soluble impurity in the periclase used is lime, and laboratory tests have shown that sufficient of it dissolves to cause almost complete deflocculation of a clay-periclase-water mixture, the clay being used in relatively small amount, less than 3 per cent. If the lime is not neutralized, an undesirably large amount of clay must be used to secure good tamping, 4 or 5 per cent being required, which, of course, introduces too much flux into the mixture. If only lime, clay and water were used, and it were desired to add acid to neutralize the lime and flocculate the clay, the acid addition would have to be made very carefully indeed, and the addition controlled with a suitable indicator, and this would become a decidedly arduous task in making up a converter lining of 10 or 15 tons. However, when using a mixture of clay, water and periclase containing lime, the problem becomes simple. All that is necessary is to add a slight excess of acid over that required to neutralize the lime. The excess acid is slowly neutralized by the periclase itself, giving a substantially neutral solution. For this reason, a slight excess of acid does no harm, but a large excess is certainly undesirable, as the magnesium sulfate formed is not a desirable binder for converter linings; it is not a tamping bond, it sets too rigidly on drying, loses most of its bonding power at a low temperature and means destroying refractory periclase to form nonrefractory magnesium sulfate.

The result of the increased plasticity due to the use of a relatively infinitesimal amount of acid is that less binder is required and the limits of the permissible liquid content are widened. This last point is important because it makes the mixing of the lining materials more foolproof. The amount of liquid necessary in making up a tamping mixture cannot be determined once and for all, as changes in temperature and in the physical qualities of the materials used necessitate changes; thus the amount of liquid to be added must be judged by the operator who makes up the mixture. The use of the very small amount of acid can eliminate to some extent the danger of spoiling a lining mixture by using too much or too little liquid, since the use of acid widens the limits of permissible liquid content of the mixture.

Grain Size

To secure greatest mechanical strength, extremes in sizes are essential; that is, very coarse and very fine grains with few, or no, intermediate

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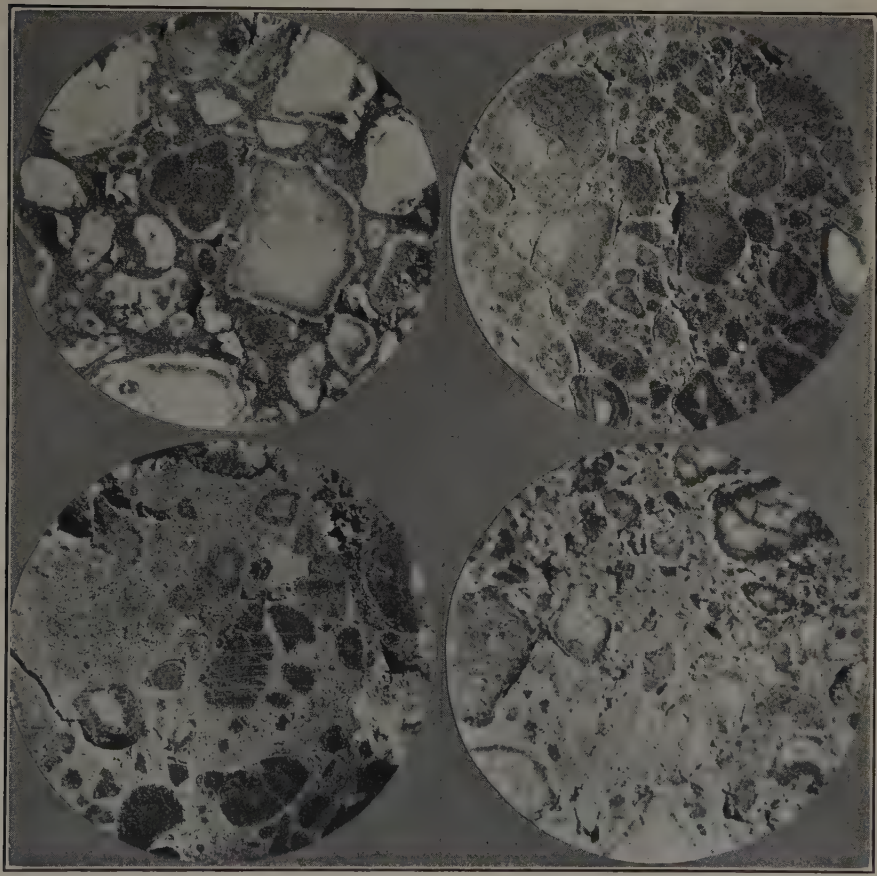

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Fig. 1.—Section of tamped periclase lining that performed well in service. Contains 16 per cent minus 100-mesh material. Shows possibility of improvement by removing some of the intermediate sizes.
Fig. 2.—Section of tamped magnesite lining that gave medium good service. Maximum grain size too small and too many intermediate sizes present. Contains 18 per cent minus 100-mesh material.
Fig. 3.—Section of tamped magnesite lining that gave poor service. Maximum grain size too small and too much fines. Contains 22 per cent minus 100-mesh material.
Fig. 4.—Section of tamped magnesite lining that gave poor service. An example of an exceedingly poor structure. Contains 27 per cent minus 100-mesh material.
Original magnification 2. Reduced about 20 per cent.

sizes. In any lining mixture the structure and resulting mechanical strength will be determined by the size that is present in sufficient

amount so that the grains will not be forced out of most compact position by smaller sizes. If the structure is to be that of 4-mesh grains, none of the smaller grain sizes should be present in sufficient amount to force the 4-mesh grains out of their most compact position. By themselves, 4-mesh grains in most compact position may have 40 per cent voids, which may be filled in with 40 per cent of minus 100-mesh periclase; which, in turn, may have 40 per cent voids. This will leave only 16 per cent voids in the mixture, and the mechanical structure will be that of 4-mesh grains, since they will still remain in contact. The 40 per cent voids in the 4-mesh material cannot be filled in with 40 per cent volume of 10-mesh grains with 40 per cent voids and the same results obtained, because the 4-mesh grains will be forced apart and the structure will have the strength and structure of the 10-mesh material. Likewise, if the minus 100-mesh material is present in sufficient quantities to force the larger grains apart, the structural strength of the lining will be that of minus 100-mesh material, regardless of other sizes that may be present. The finer sizes show greater sensitivity to thermal shock, greater fluxibility, greater shrinkage, therefore variation of only a small percentage of the finer sizes may mean complete failure of the lining when it is put into service. Greatest density is obtained by having all sizes present in proper proportions, but the much more important mechanical strength is obtained by mixtures of very coarse and very fine material. Variation in each extreme is permissible, but a band of the intermediate sizes should be missing, this band moving either way as the predominating size is coarser or finer. One reason this sort of mixture gives greatest mechanical strength is that the large grains seem to be lubricated by the fines, and to have slipped into most compact position when tamped, whereas when all sizes of grain are present the larger seem to be prevented from taking the most compact arrangement unless an excessive amount of very fine material is used. This, however, decreases the mechanical strength, under operating conditions, to a point where it ceases to be useful for refractory purposes.

Water in the Mix

Importance of the question of addition of water to the mix was not recognized for a considerable time. With too much water the mixture was rubbery under the hammer and tamped unsatisfactorily. The performance, during service, of the linings that were tamped with too much water was exactly the same as those in which too many fines were used. Mixtures made with molasses only, without water were stiff, tamped solidly and gave very satisfactory service, and it was soon found that if only a small amount of water was used with the clay mixture equally good results were obtained.

The periclase that was finally selected as best suited to this work has the analysis shown in Table 1.

TABLE 1.—*Analyses of Periclase and Fireclay Finally Selected*

ANALYSIS OF PERICLASE, PER CENT

	Ignition Loss	SiO_2	R_2O_3	CaO	MgO
Low	None	4.62	1.38	2.70	91.58
High	0.18	2.78	0.70	1.67	94.08
Average	0.10	3.80	1.10	2.00	92.52

SCREEN ANALYSIS OF PERICLASE AS RECEIVED IN CARLOAD LOTS, PER CENT

+3	+4	+6	+8	+20	+48	+200	−200
18.7	37.3	52.9	68.1	95.3	99.3	99.8	0.2

ANALYSIS OF FIRECLAY, PER CENT

SiO_2	Al_2O_3	Fe_2O_3	CaO	MgO	Ignition Loss
59.0	26.0	3.4	1.1	1.6	9.5

DESIGN AND CONSTRUCTION

The equipment used for the acid linings during 1925 and 1926 was found to be suitable for the work of tamping the monolithic periclase linings into the converters. This equipment consisted of a "wet pan" of 84-in. diameter and ring 13 in. high. The mill was fed from two overhead bins, one of 30 tons capacity holding the periclase, and one of 6 tons capacity holding the fireclay. The discharge from the periclase bin was into a measuring box calibrated for 350 lb. of periclase, and discharged by an air ram into the mill. The fireclay, if used, was discharged into a container and weighed. Pulverized periclase was in sacks and was weighed as used. Molasses and acid were measured, as desired, into the pan. The predetermined charge, when mixed, was discharged through a swinging gate on to a steel platform and from there was shoveled into the converter ready for tamping. The tamping machine consists of a swinging jib or crane attached to a crane column. This jib was 17 ft. long over all, and was equipped with a ball-bearing trolley mounted on rails, designed to allow the tamping cylinder to travel the full length of the jib. The hinge attaching the jib to the crane column gave the other plane of movement, allowing the hammer to be placed wherever desired. The air cylinder was an Ingersoll-Rand rock-drill machine suspended to the trolley by a worm screw 77 in. long and 4½-in. diameter, which gave the third vertical plane of movement. Air was delivered to the hammer at about 90 lb. per square inch. A cast-iron shoe plate, 9 in. square, was fastened to the steel chuck stem and shoe with countersunk bolts.

The tamping form used to retain the lining mixture was approximately semicircular, 9 ft. 6 in. in diameter at the top and 9 ft. at the bottom. This was placed in the cleaned-out shell, so that it was 20 to 23 in.

from the shell at the center of the tuyere line and 19 in. at the top. The ends were fitted with wings, so that the opening between the space to be lined and the old "belly" could be closed during tamping operations. The form was held in place during the tamping by six pieces of 2 by 4-ft. lumber wedged against the shell.

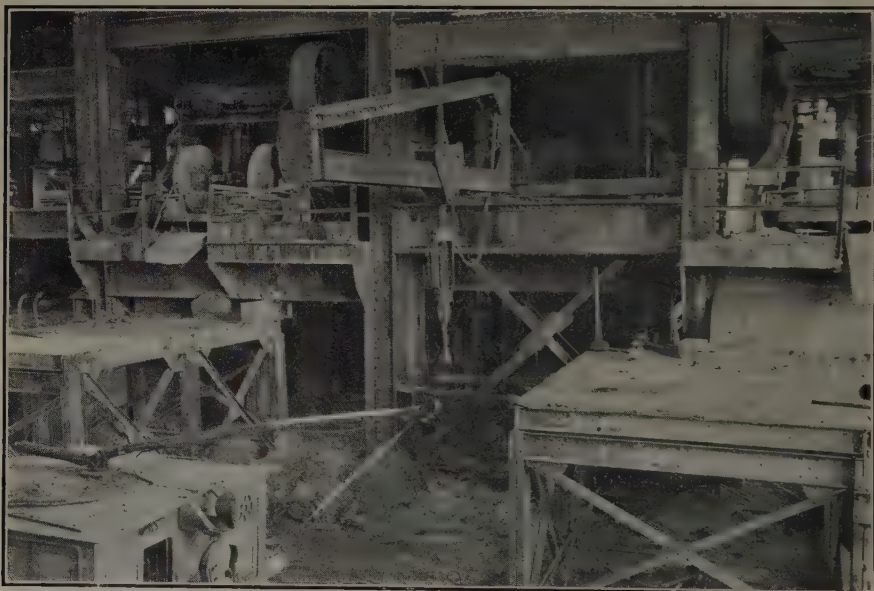

FIG. 5.—MACHINERY USED BY UNITED VERDE COPPER CO. FOR MIXING AND TAMPING PERICLASE CONVERTER LININGS.

LINING PRACTICE

The practice of lining 12-ft. Great Falls converter shells is as follows: After the converter to be tamped has been "washed out" and sufficiently cooled, the wedges and pins are pulled from around the cap and the cap is lifted off. The old lining is then cut cut with a paving breaker. All slag, matte and unused old periclase lining is cut out to a point about 10 in. below the tuyere line, at which point a good foundation is usually found on which to rest the tamping form and to provide a suitable shelf for the new lining. The sides are cut back as far as the form extends. All the old lining not impregnated with matte and slag is sorted out and saved, to be used again in the cap lining and in the seal between the shell and cap. The floor or front of the shell very seldom requires patching. At the tuyere line and about 18 in. above, the old lining is cut out to expose the steel shell. The old tuyere pipes are loosened and pulled out, and the shell is placed in position ready to tamp. The cap is cut out, cleaned up and made ready to put back into place.

If it has been decided to line the converter with the clay and periclase mixture, the procedure is as follows: 40 lb. of fireclay and 135 lb. of periclase fines (product of a Raymond mill and practically all through 200 mesh) are placed in the wet pan, wet slightly with water and mixed for about five minutes. While the mill is still revolving, three boxes of 350 lb. each of coarse periclase are pushed on to the mixing pan and a little more water is added. Not more than 5 gal. of water should be used in each batch. As the coarse material is being mixed, 4½ lb. of sulfuric acid is added. The mill is then run until the ingredients are thoroughly mixed, or until each coarse grain is coated with an envelope of clay and fines—this takes but a few minutes—then the door of the mill is opened and the mixed batch is run on to a steel platform. It takes about one and one-half mill batches to make one tamping batch, therefore one and one-half batches are mixed and shoveled into the converter behind the form. This mixed material is leveled up and the shell is ready for tamping.

If a molasses-periclase lining is to be used, the following procedure is followed: The wheels of the mill are raised about one inch above the pan; 175 lb. of fine periclase and 5 gal. of blackstrap molasses are placed in the pan with very little water, and the mill is started. Three boxes of coarse periclase are pushed into the pan immediately, 4½ lb. of acid added, and the mill run just long enough to mix the ingredients; no grinding is wanted. The water added to this mix must not exceed one gallon. As in the clay mix, this is run on to the platform, another half batch added and the mixture is shoveled into the shell ready for the tamping operation.

When the mixture is prepared and in the shell the tamping machine is swung into place, the shoe screwed down until it rests on the mix, then turned down 2 in. more, and the hammer is started. During the tamping operation the shoe is turned down as the mixture becomes more tightly tamped. If the shoe is too high the machine will bounce and if too low the piston will work too fast. About 45 min. of tamping per layer is necessary to secure proper density. When the lining is up to the tuyere level, the new tuyere pipes are installed and accurately lined. The pipes are tightly tamped with asbestos between the shell and pipe, the spanner nuts are tightened and 1½-in. pipes are inserted in the 2-in. tuyere pipes to protect them from being distorted by the subsequent tamping operation. Then, one-half batch of the mix is shoveled around the tuyere pipes and tamped very carefully both by hand and by air machine between and around the pipes. After the tuyere line is properly tamped and the pipes covered, the regular one and one-half batch lots are tamped approximately 45 min. each until the lining is finished to the top of the shell. After the lining has been completed to the shell rim the tamping machine is swung to one side, the form is released from its braces and carefully pulled out by the crane. The 1½-in. protecting

pipes are removed from the tuyere pipes and the tuyere holes cleaned out. Fireclay is then ground in the mill and a layer about 2 in. thick is placed around the flange of the converter. The cap is then set in position and the pins are dropped into place but not tightened. A few batches of old used periclase and some clay and water are ground to a sticky paste and plastered by hand into the junction between the new lining and the cap, and the top is then tightened down by driving in the wedges. The converter is now ready for drying, first by a wood fire and then by an oil fire. This is not done at once, however, as new linings are not dried until the shell is to be put into service.

Great care must be exercised in the drying-out process. A slow wood fire is first started and kept burning for 8 hr. and then a small oil flame is used. Practically all moisture must have been expelled before much heat is applied, and when no more steam is seen at the vent holes in the shell the flame is increased and the whole lining is heated until the steel shell is hot to the hand. The lining has now received its initial set, and if properly mixed and tamped will not spall. The shell is now transferred to the stand where it is immediately charged with matte and blown.

An average of 13.61 tons of coarse periclase, 1.35 tons of fine periclase, 0.577 tons of fireclay and 94 lb. of sulfuric acid is used in a periclase-clay lining. Using molasses as a binder, 12.22 tons of coarse periclase, 1.88 tons of fine periclase, 73.65 gal. of molasses and 79.8 lb. of sulfuric acid is the average of 30 shells.

Costs

To arrive at a divisor to determine the unit cost of different types of linings, it was necessary to establish an arbitrary relationship between a ton of iron oxidized and a ton of copper finished in any given shell. Owing to the grade of the matte converted at United Verde, no shell ever blows to white metal all the metal it finishes to copper. Transfers

Table 2.—*Comparative Costs of Converter Linings*

Lining	Number of Shells	Days of Campaign	Total Cost of Lining	Units Iron Oxidized Plus Equivalent Copper	Unit Cost
Brick[a]	16	22.75	$1029.13	1969	$0.523
Periclase and boric acid	6	18.70	945.70	1674	0.564
Periclase, molasses or clay, but no added fines	16	21.10	859.92	1750	0.491
Periclase, fine periclase, molasses and H_2SO_4	30	34.07	908.80	2427	0.374
Periclase, fine periclase, fireclay and H_2SO_4	15	35.40	895.52	2442	0.367

[a] The bricks were 15-in. radial bricks of a standard manufacture.

of white metal from two or three shells are combined in one converter to make a charge sufficiently large to blow to blister. A study of the time involved in the different operations led to the arbitrary method of adding to the tons of iron oxidized one and one-half times the tons of copper finished and using the sum, expressed as "iron plus equivalent copper" as the divisor into the total cost. No accuracy is claimed for this method, but as the same method was used for all different types of linings, and as each shell finished approximately the same proportion of the total copper as it oxidized iron, this method gives a fair indication of relative costs. Table 2 gives total and unit costs of five different linings.

Conclusions

In order to still further improve converter costs it has been decided to screen the periclase as received in future, to pulverize undesirable sizes—say 8-mesh and smaller—and use the pulverized material to mix with the coarse. The idea of continuous tamping also will be studied. It is noticeable that the tops of the layers are much more solidly tamped than the bottom, and if equipment could be installed so that the whole lining mix could be prepared at one time, put into overhead bins and fed continuously into the form, and be continuously tamped, it is probable that a better lining would result.

The fact that this experimental work has been done in the converter aisle without any interference with operations has tended to materially slow the interpretation of results. A shell tamped with a new mixture, or different proportions of a given mix, might not be put into service and its lining burned out until two months had elapsed after tamping, and any real conclusions as to whether the change was beneficial or not had to wait a much longer time than that. If the change indicated that a seeming improvement had been effected, at least four more shells, tamped in the same manner were used to check the results obtained on the first lining.

Acknowledgments

The author gratefully acknowledges many valuable and timely suggestions made by Mr. Max Y. Seaton, of the Sierra Magnesite Co., during the prosecution of the work described.

DISCUSSION

M. W. Krejci, Worcester, N. Y. (written discussion).—Mr. Parsons' paper reflects the progressive spirit of the staff at the United Verde plant. The results speak for themselves, as evidenced by a 30 per cent reduction in lining costs, when compared to magnesite brick. The only suggestion I would offer, in case it has not been already tried there, is to charge to the freshly lined converter, after it has been thoroughly dried, with hot converter slag, and allow it to remain there for about 20 minutes. Before doing this, the tuyeres should be sealed on the inside of the converter, with

clay, so that the slag will not enter the tuyeres. The slag will coat the periclase lining, and more thoroughly set and dry it. The converter should be charged within 30 to 45 min. after the slag is removed.

The work that has been done is another example of finding the silver lining in the cloud. The lining equipment used for this work was installed during the time when the United Verde was enjoined from using the magnesite linings, obviously at some expense. Now the same equipment is being used to reduce costs. The United Verde is to be congratulated on being able to reduce its converter costs, particularly at this time, when the price of copper is so low. It is an example that other research organizations might follow.

The Noranda Smelter

By W. B. Boggs* and J. N. Anderson,† Noranda, Quebec

An account of the operation of the Noranda smelter covering the period from December, 1927, when the smelter started, until December, 1929, was given in a paper published by the Canadian Institute of Mining and Metallurgy early in 1930.[1] The object of the present paper is to review matters of interest in the original paper and to describe the present construction and operation of the smelter, with particular reference to the reverberatory-furnace practice.

The Noranda smelter was built by Noranda Mines, Ltd., from the designs of A. E. Wheeler, consulting metallurgist of New York, for the reduction of ores and concentrates from the Horne mine and to treat, on a custom basis, ores and concentrates from other mines in the Rouyn district. The smelter is at Noranda, about 330 miles directly north of Toronto. Construction work was started in June, 1926, and copper was first produced in December, 1927.

The smelter was built throughout as two duplicate units, each with a rated capacity of 500 tons of ore per day. Operations were started with one unit and after several months it was found that one unit alone could smelt 1000 tons of ore a day. At that time there was only one hoisting shaft, No. 3, and its hoisting capacity and the capacity of the original crushing plant were insufficient to supply ore for the two units at the increased tonnage. Accordingly No. 4 shaft was sunk and a new crushing plant provided to handle the ore hoisted there. Operating two units at capacity also required the installation of two more converters to treat the additional quantity of matte produced. This work was completed in the latter part of November, 1929, and since then both smelter units have been operating continuously.

In 1928, a mill to treat 500 tons a day of concentrating ore was completed. Since then the concentrator capacity has been increased twice, and at present 2000 tons a day of ore are being concentrated.

The layout of the surface plant of the mine, concentrator and smelter is shown in Fig. 1 and the general plan in Fig. 2. The smelter is shown in more detail in Fig. 3. The original hoisting shaft, No. 3, is now used only for handling men, supplies and waste rock, and the original crushing

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* Smelter Superintendent, Noranda Mines, Ltd.

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[1] W. B. Boggs and J. N. Anderson: The Noranda Smelter. *Canadian Min. & Met. Bull.* (March, 1930).

FIG. 1.—AERIAL PHOTOGRAPH OF SURFACE PLANT.

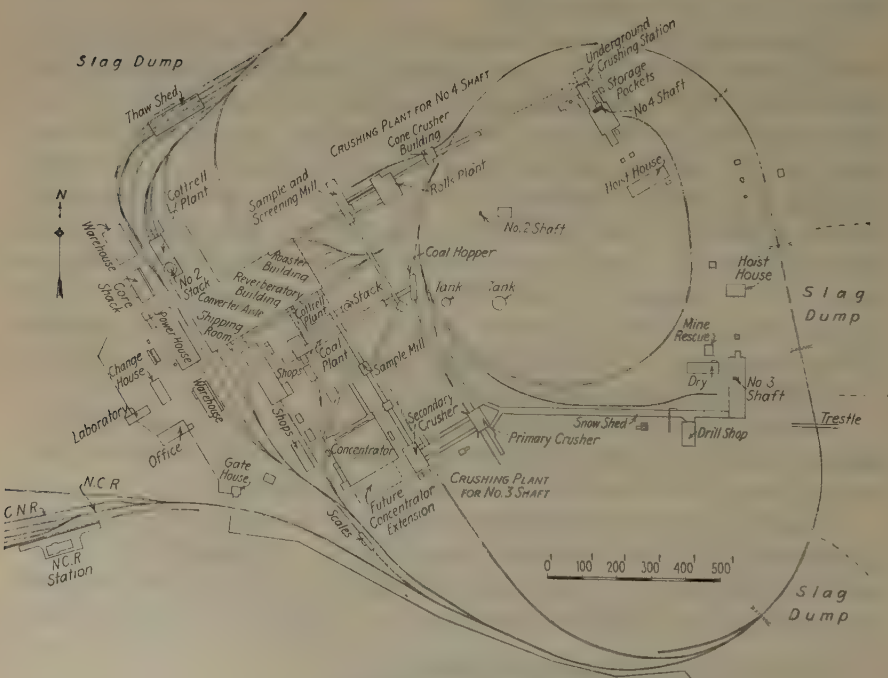


FIG. 2.—GENERAL SURFACE PLAN.

plant handles only smelter reverts, custom ore and waste rock used for town road construction. All smelting, concentrating and fluxing ores are hoisted in No. 4 shaft and crushed in the new, or No. 2, crushing plant.

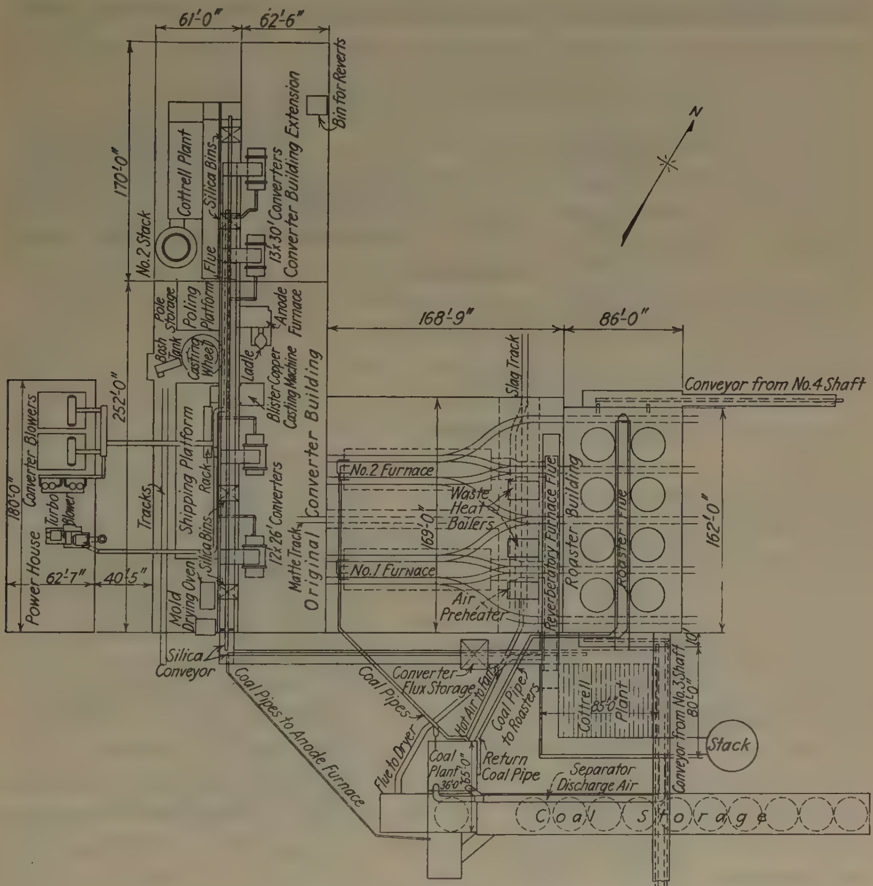


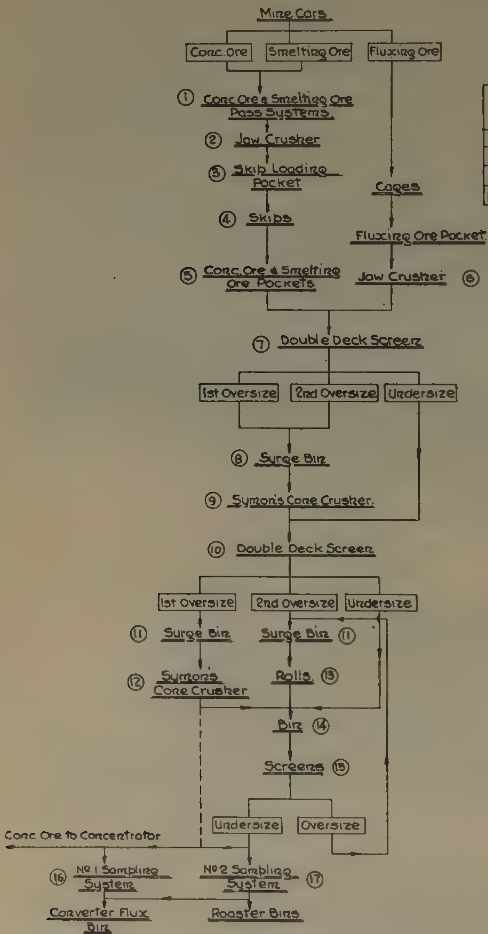
FIG. 3.—LAYOUT OF SMELTER.

In a reverberatory-furnace smelting plant the total tonnage smelted is limited by the quantity of charge that can be smelted in the furnaces. Early in 1928, the furnace capacity was about 700 tons of solid charge per furnace day; by improvements in furnace construction and operation the capacity has been gradually increased to 1267 tons of solid charge per furnace day in December, 1933, resulting in a decrease in the cost per ton.

CRUSHING PLANT

A flow sheet showing the handling of the ore from the mine through the crushing plant to the smelter and concentrator is given in Fig. 4. Separate ore-pass systems for smelting ore and concentrating ore are provided in the mine, giving storage for about 5000 tons of each class of

ore. Each system terminates in a storage pocket above a crushing station on the 1200-ft. level, where the ore is crushed in a jaw crusher and carried by a belt conveyor to a skip loading pocket. At the surface there are three separate storage pockets, one for each class of ore, extending to the



Tonnages and Screen Analyses of Final Crushing Plant Product

	Tons per hour	Screen Analyses				
		% on $\frac{1}{2}$ in	% thru $\frac{1}{2}$ in. on 3 mesh	% thru 3 mesh on 6 mesh	% thru 6 mesh on 10 mesh	% thru 10 mesh
Smelting Ore	350		8.5	33.8	18.8	38.9
Concentrating Ore	400		10.5	30.3	16.0	43.2
Fluxing Ore to Roasters	240		18.0	35.0	16.0	31.0
Fluxing Ore to Converters	325	13.7	50.5	20.1	5.4	9.3.

LEGEND

① Concentrating & Smelting Ore Pass Systems- 500 to 1200 ft. levels - Cap. 5000 tons each
② 1- 36"x 48" Buchanan Jaw Crusher- 1200 ft. level
③ Skip Loading Pocket-1200 to 1300 ft levels-cap 1500 T.
④ 6 Ton Skips.
⑤ Ore Pockets from surface to 125 ft level Cap. Conc. Ore & Smelting Pockets- 2000 Tons ea. Cap. Fluxing Ore Pocket - 1000 Tons
⑥ 1- 36"x 48" Buchanan Jaw Crusher- 125 ft level
⑦ 1- 4'x 6' Double Deck Niagara Screen Openings top deck 4"x 4"- openings bottom deck 1"x 2"
⑧ 15 Ton Surge Bin.
⑨ 7 ft. Symons Cone Crusher-setting 1"
⑩ 1-5'x 8' Double Deck Niagara Screen openings top deck 3/4"x 2", openings bott deck 3/8"x 4"
⑪ 100 Ton Surge Bins.
⑫ 5½ ft. Symons Cone Crusher
⑬ 78"x 28" Traylor Rolls
⑭ 75 Ton Bin.
⑮ 5- 48"x 77" Traylor Screens & 2-48"x 96" Niagara Screens.- openings 1/4"x 2"
⑯⑰ Sampling Systems. see detailed flow sheet

Fig. 4.—Flow sheet of crushing plant.

125-ft. level. Smelting ore and concentrating ore are discharged from the skips into their respective pockets, while the fluxing ore is hoisted in cars and dumped into the fluxing-ore pocket by means of a rotary tipple. At the 125-ft. level there is another jaw crusher discharging on to an inclined belt conveyor, which carries the ore to the surface crushing plant. The ores are drawn from the pockets by means of Ross feeders; bypasses allow the smelting ore and concentrating ore that have already been crushed to be discharged from the pockets direct to the belt conveyor without passing through the jaw crusher.

Details of the surface crushing plant are shown in Fig. 4. Several changes have been made in the original construction, the most important of which are as follows: (1) providing a surge bin above the 7-ft. cone crusher; (2) using two 200-hp. motors on the rolls instead of two 100-hp. motors, and increasing the speed of the rolls from 100 to 120 r.p.m.; and (3) installing a 5½-ft. cone crusher in parallel with the rolls to take part of the product of the 7-ft. cone crusher. The object of the first of these changes was to provide for uniform feeding of the 7-ft. cone crusher and thus to prevent damage by overloading. By making the second change the capacity of the rolls was increased. The 5½-ft. cone crusher was part of the equipment of the old crushing plant and became available when all the ore was hoisted in No. 4 shaft.

The product of the 7-ft. cone crusher is sent to a double-decked screen, and the oversize of the top deck is sent to the surge bin above the 5½-ft. cone crusher; the oversize of the bottom deck is sent to the surge bin above the rolls; and the undersize of the bottom deck bypasses the rolls. The products of the 5½-ft. cone and the rolls go to the screen house. The smelting ore, concentrating ore and the part of the fluxing ore that is fed to the roasters are crushed in closed circuit, while the converter flux, which must be coarser, is crushed in open circuit. For crushing converter flux provision has been made, so that the joint product of the 5½-ft. cone crusher and the rolls may bypass the screens, or the unscreened product of the 5½-ft. cone crusher alone may be sent to the converter-flux storage bin, while the rolls product is screened and the screen undersize sent to the roaster-flux storage bins.

All smelting and fluxing ores are sampled by a mill consisting of four Vezin samplers in series together with a set of rolls and a sample grinder, and are weighed on Merrick weightometers. Concentrating ore is weighed and sampled at the concentrator. The crushing rate and screen sizes of the final crushing-plant product from the three classes of ore are given in the flow sheet in Fig. 4.

The fineness of the roaster feed is of great importance in both the roasting and the reverberatory furnace operations and is controlled by changing the mesh size of one or more of the screens.

The Smelter

When the design of the smelter was decided on, it was known that there was ore in the mine that would have to be concentrated, and that there was the probability of finding more ore of similar grade. Therefore roasters and reverberatory furnaces were the choice rather than blast furnaces.

No bedding system was provided because of the difficulty of storing and handling large tonnages of ores and concentrates in the severe winter weather. Instead, a system of storage bins over the roasters was

installed, consisting of three storage bins for each roaster, together with two spare bins for special material and a number of bypasses, by means of which material such as fluxing ore for the furnace fettling and reverts to be smelted could be sent direct to the reverberatory-furnace feed floor without going through the roasters. The three bins for each roaster are used for the separate storage of smelting ore, fluxing ore and concentrates, giving a storage capacity of 3200 tons, 2100 tons and 1120 tons, respectively. Material is fed from the bins to the top hearth of the roasters by means of pan feeders provided with adjustable gates. In the original construction the converter-flux storage system was inadequate and the handling facilities unsatisfactory, therefore in 1929 a 500-ton storage bin was built, together with a 60-ton bin over each of the four converters.

The roasters, reverberatory furnaces and converters were all placed under one roof, with no dividing walls, so that the maximum effect of the heat from the various processes would be utilized in keeping the interior at a comfortable temperature during the cold weather. This was found to be a mistake because gas from the converter aisle drifted into the reverberatory and roaster buildings and made bad working conditions. Dividing walls were built, so that now, in effect, the three buildings are separate.

There are eight roasters in two rows (Fig. 3), at right angles to the long axes of the reverberatory furnaces, which necessitates considerable switching of the trains when drawing the calcine from the roasters. This could have been avoided by placing the roasters in a double row parallel to the long axes of the reverberatory furnaces, with a single track running under calcine bins placed on the center line of the two rows of roasters.

The roasters are Wedge furnaces, 25 ft. in diameter, with an external feed or drying hearth and seven internal hearths. There are two air-cooled arms on each hearth, one arm carrying 10 and the other 11 rabble blades. Each roaster is driven by a 15-hp. motor belt-connected to a speed reducer, and the drive is provided with a shear-pin arrangement to prevent breaking of the arms in case of overload or jamming in the roaster. Each roaster has two gas outlets 4 ft. in diameter, connected to the roaster flue. The latter is a steel balloon flue with a cross-sectional area of 130 sq. ft., leading to the header flue of the roaster Cottrell plant. The balloon flue dust is drawn off through slide gates into pipes which discharge into the calcine hoppers.

The floor below the roasters is an extension of the reverberatory-furnace feed floor. The calcined ore falls into hoppers above this floor and is drawn to the furnace bins in cars hauled by storage-battery locomotives. The furnace feed bins extend along both sides of the furnace where the calcine cars are unloaded, as required, by properly spotting them over the furnace bins.

The two reverberatory furnaces are side fed and fired with pulverized coal; their construction is shown in Fig. 12. The furnace slag is skimmed into pots of 20 tons capacity. Originally the pots were drawn by two 20-ton storage-battery locomotives, which proved unsatisfactory for continuous heavy service and were converted to trolley motors. The reverberatory-furnace matte is tapped into cast-steel ladles of 12 tons capacity. The ladle is placed on a transfer car, which runs in a tunnel between the two furnaces below the level of the furnace floor. When filled the ladle is drawn out to the converter aisle by an electrically driven car puller and is then picked up by the cranes that serve the converters.

Originally the converter building had two Peirce-Smith converters, 12 ft. in diameter and 26 ft. long; two straight-line casting machines for making blister bars; two 40-ton d.c. cranes, and one 10-ton crane in the shipping aisle behind the converters. In 1929 the converter building was extended to make room for two more Peirce-Smith converters, 13 by 30 ft. in size.

In order to prevent the molten bath in the converters from filling the tuyeres in case of sudden failure of the converter air supply, a storage-battery set is connected to the converter tilting motors, so that if the power goes off the blower motors the converters are automatically turned until the tuyeres are clear.

Until the end of 1930 all the copper produced was shipped as blister bars for refining in the United States. In 1930 Canadian Copper Refiners, Ltd. built a refinery at Montreal[2] for refining the copper from Noranda and Hudson Bay Mining & Smelting Co., and it was decided that all Noranda copper could be shipped as anodes. Accordingly, a 150-ton anode furnace and a Walker casting wheel were built,[3] and since early in 1931 all the copper produced has been shipped to Montreal in the form of anodes.

Preheater and Waste-heat Boilers

Waste-heat boilers to generate steam for power are usually included in the design of reverberatory matting furnaces in copper smelters. This was not done at Noranda because comparatively cheap hydro-electric power was available and it was thought that the heat in the waste furnace gases could be utilized and a considerable saving in fuel effected by means of a heat recuperator to provide hot air for the burning of the pulverized coal.

The original design of the heat recuperator or air preheater is shown in Fig. 5. Some changes have been made but the general principle remains the same. It consists of an enclosed chamber in which are a

[2] See page 352.

[3] See page 331.

number of vertical tubes through which the gases pass while the air to be heated is circulated around the tubes. There are 18 tubes, 24 in. in diameter and 20 ft. high, set in three rows of six each. Circulation is counter-current; that is, the gases pass upwards through the tubes while the air is blown in at the top of the chamber by a fan, circulates around the tubes and discharges at the bottom into a duct leading to the furnace. In the original design there were six horizontal air passes in series and baffles were provided inside the tubes in order that the hot gases might have maximum contact with the tubes. The tubes and baffles were made of "Hybnickel," a patented heat-resisting chrome-nickel-iron alloy.

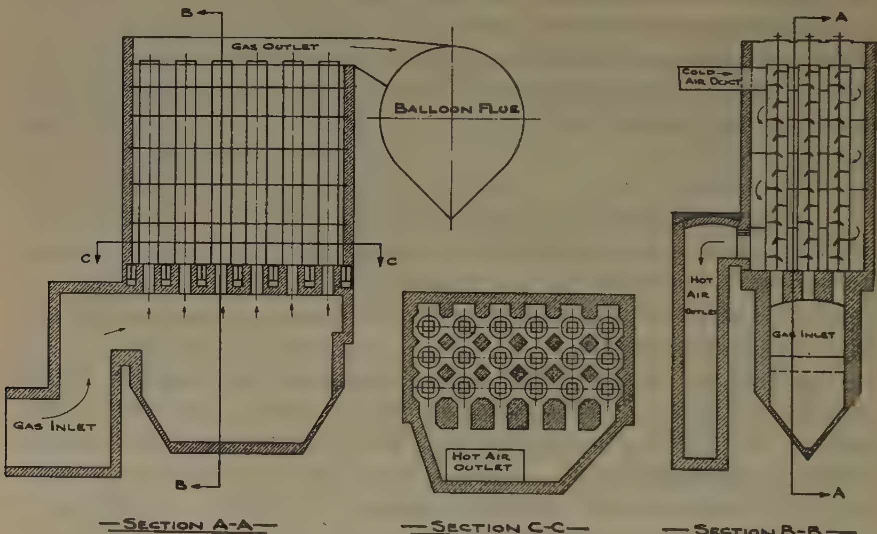


Fig. 5.—Original design of air preheater.

The first trouble that developed was caused by the ash and dust in the gases depositing on the baffles and building up in the brick "chimneys" at the bottoms of the tubes. As no provision had been made for properly cleaning the tubes and the baffles, it was found necessary, after a few days, to remove the baffles entirely, the tubes and "chimneys" being kept clean by an improvised cleaning rod.

During the first week of operation the temperature of the preheated air was as high as 1000° F. but after the baffles were removed this temperature could not be maintained and dropped to about 350° F. This temperature remained fairly constant for several months and then gradually dropped. At this time it was discovered that the system was not airtight; tests showed that the quantity of air leaving the preheater was less than that delivered to it by the fan. After the first six months of operation, and until No. 1 furnace was shut down for general repairs in January, 1929, the temperature of the preheated air was about 250° F. It was found

that several of the lower sections of the Hybnickel tubes had cracked and that some of the Hybnickel supporting beams had also failed. These beams had been placed at the bottom of the preheater to carry the weight of the tubes and had been exposed to the direct heat of the gases entering the preheater. The failure of the tubes and beams may have been due to overheating, or to stresses set up by rapid heating and cooling during operation. The surfaces of the tubes and beams showed no signs of corrosion.

It was decided to make some alterations in the construction, with the object of preventing the recurrence of such troubles. The changes made

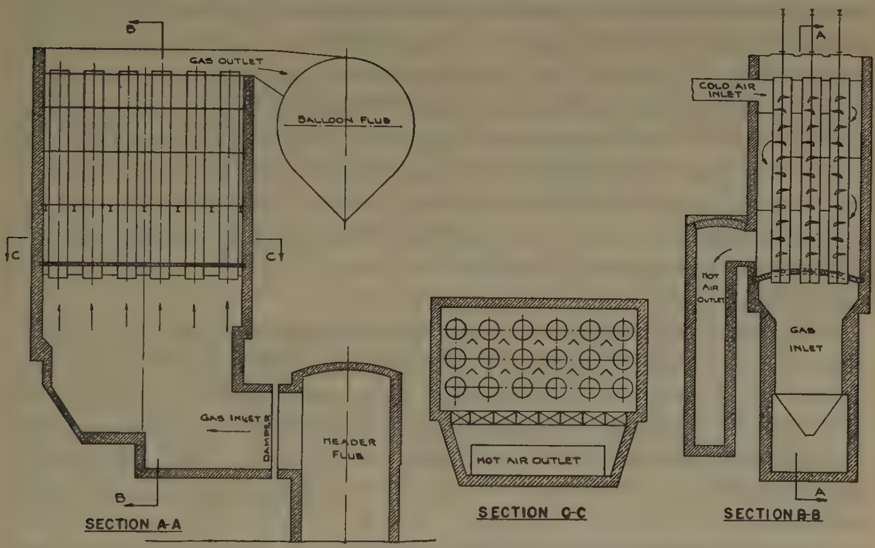


FIG. 6.—PREHEATER AFTER ALTERATION.

are shown in Fig. 6. The baffles were supported from overhead rods in such a way that they could be freely rotated in order to clean the baffles and tubes. The supporting beams were placed higher in the preheater, with the lower sections of the tubes suspended from them. An arch of Carbofrax brick was built below the beams to protect them from the direct heat of the furnace gas, Carbofrax being chosen because of its high thermal conductivity and low coefficient of expansion. This arch was so placed that the bottom sections of the tubes protruded a short distance through it, which made it possible to remove the brick "chimneys" that previously had given trouble. The number of air passes was reduced from six to four and their areas were so designed that the velocity of the air passing through the preheater would remain constant as its volume increased with the increase in temperature.

The reconstructed preheater was started in September, 1929, and operated for two months, when it was shut down for repairs. The Carbofrax arch was in perfect condition, but there was considerable

leakage through the seal between the tubes and the arch. A new form of seal was devised consisting of a tapered Hybnickel ring set around the tubes and filled with refractory cement. Many of the bolts that attached the baffles to the supporting rods had sheared off and these were replaced with bolts of a material less liable to failure at high temperature. One of the supporting beams had cracked and was replaced.

After these changes had been made the preheater was started again and has been operating continuously ever since, except during furnace shutdowns and for short inspection periods, but for two reasons it has been found impossible to obtain preheated air over 300° F. in temperature for any length of time. First, the tubes cannot be kept free from the dust and ash from the furnace gases. Despite the most careful attention the baffles become clogged with semifused dust and ash and cannot be operated, with the result that the tubes fill up and no gas passes through them. If the baffles are removed the inside surface of the tubes becomes coated with dust and ash, thus reducing the transfer of the heat. Second, it has been found impossible to prevent leakage from the air to the gas side; this leakage reduces the quantity of gas passing through the preheater and consequently reduces the temperature to which the air is heated.

At present the gas inlet and outlet temperatures are 1400° and 800° F., while the air inlet temperature varies from −20° to 80° F. and the heated air is approximately 300° F. No exact figures are available for the volume of gas and air passing through the preheater, but it is known that only a small part of the gases from one furnace can be used and that about 40 per cent of the inlet air is lost by leakage from the air to the gas side.

Since 1929 firing practice on the reverberatory furnaces has changed so that now only a small percentage of the air required for combustion can be sent from the preheater directly to the furnace. With the unit system of coal firing now being used, about 35 per cent of the combustion air is used as primary air for classifying the coal and conveying it to the burners. From 40 to 45 per cent of the combustion air is supplied to the burners at a pressure of 5 lb. per square inch from the converter blowing engines. The result is that only a small percentage of the total combustion air could be supplied as high-temperature secondary air. The hot air from the preheater is utilized by sending it to the coal plant and introducing it into the coal classifiers in such quantity that the temperature of the mixture of pulverized coal and air leaving the classifiers is about 150° F. This temperature is considered the maximum possible for safe operation, on account of the danger from fire or explosion with the high-sulfur coal being used.

Thus, the main function of the preheater at present is to supply warm primary air; a small amount is introduced under the burners of one

furnace as shown in Fig. 11. On the other furnace some of the hot air from the cooling of the roaster arms is similarly used.

The preheater is a failure in its designed purpose of utilizing the heat in the waste furnace gases to supply high-temperature combustion air. The failure has been due partly to design but mostly to the dust and ash in the furnace gases. In its present function of supplying warm primary air the preheater undoubtedly gives higher flame temperature and a better fuel ratio than could be obtained without it, but this could be done by the use of apparatus less costly to construct and maintain. Unless purchased power is very cheap, the authors believe that a well designed waste-heat boiler installation for generating steam for power would show greater savings than the type of preheater now used at Noranda.

Steam for plant-heating purposes was supplied originally by a coal-fired boiler but this proved to be too small and in 1929 a 750-hp. Kidwell waste-heat boiler was installed. The furnace flue arrangement was such that only the gases from No. 2 furnace could be sent to the boiler, with the result that when a shutdown for repairs to the furnace or the boiler occurred in the winter months considerable damage to the heating system resulted, as well as inconvenience from lack of steam heat. The quantity of steam required and the necessity for a continuous supply were further increased by the use of process steam in the concentrator. Accordingly, early in 1933 a second waste-heat boiler similar to the first was installed; at the same time the two outlet flues on each furnace were replaced by a single outlet flue leading to a new header flue from which the gases from the two furnaces could be sent to either or both boilers and the preheater. Since the boilers and the preheater could not take all the gases from the furnaces, provision was made for bypassing the excess gases into the furnace balloon flue. With this arrangement a constant supply of steam is available and it will be possible to make waste-heat steam for power should this be considered advisable at any time in the future.

Coal Plant

The flow sheet of the present coal plant is shown in Fig. 7. Briefly, it consists of crushing and storing facilities for raw coal, a rotary drier, a unit pulverizer for each furnace together with a spare pulverizer, which can be used for either furnace, and a separate storage bin and a small ball mill for supplying pulverized coal to the anode furnace.

As originally built, the coal plant was designed to use the indirect or storage system, but after considerable trouble, which culminated in a serious explosion, the system was converted to the direct or unit system. A description of the old system and an account of the trouble experienced with it might be of interest in showing the necessity for careful and correct design of pulverized coal plants.

There were two pulverizers and classifiers and an exhauster fan drew the pulverized coal from each classifier and discharged it into a large circular storage tank through openings tangential to the circumference. The coal was drawn off from the bottom of the tank by feed screws and blown through pipes to the furnace burners by a high-pressure blower. Thus, the tank functioned both as a cyclone collector and as a storage bin for pulverized coal. The capacity of the tank was 125 tons and

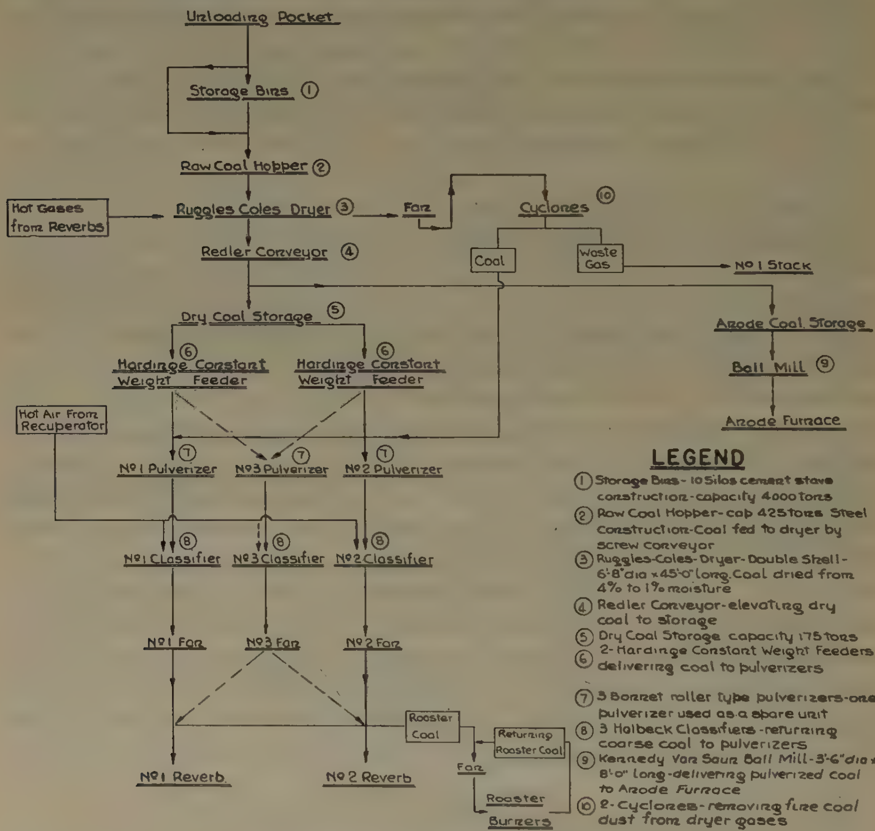


Fig. 7.—Flow sheet of coal plant.

ordinarily from 25 to 40 tons was carried in storage, the pulverizers being operated intermittently.

In September, 1928, Nova Scotia coal was substituted for Pennsylvania, and probably owing to the high sulfur content of the former (3.5 per cent) several fires occurred in the pulverized-coal storage tank. In October, 1928, an explosion and fire in the storage tank caused minor damage and smelter operations were interrupted for several days because of difficulty in extinguishing and disposing of the burning pulverized coal. Investigation showed that the explosion and the resulting fire had been caused by smoldering pulverized coal that had collected in flat places in

the pipes from the exhauster fans to the storage tank. The piping was altered to eliminate these flats and additional safety features such as clean-out doors and explosion vents were provided.

In June, 1929, another explosion occurred, which wrecked the pulverized-coal storage tank and the coal-feeding mechanism. In order to resume operations without an excessive loss of time the use of the storage tank and feed screws were dispensed with by connecting the discharge of the classifier exhauster fan to the intake of the furnace blower. This change converted the former storage system into the direct or unit system. After some weeks of operation, it was decided to double the speed of the exhauster fan and discontinue the use of the furnace blower. A third pulverizer, classifier and exhauster were installed, to be used on either furnace in case of a shutdown on either of the original pulverizers.

The explosion happened at midnight just after pulverizing for the night shift had started. One pulverizer had been running for a few minutes and when the second one was started a slight explosion occurred in the second pulverizer; this was followed immediately by a terrific explosion that blew off the top and burst the sides and bottom of the pulverized-coal tank. There were about 20 tons of pulverized coal in the tank and this took fire and burned until it was entirely consumed. Although considerable damage was done to the plant, the operators escaped with only slight injuries.

The extremely violent nature of the explosion was due to the large volume of the storage tank and to its being filled with pulverized coal and air from the operation of the first pulverizer. The primary explosion was slight because of the comparatively small volume of the pulverizer and the classifier, but it was the flame from the first explosion that traveled through the pipes and ignited the coal and air mixture in the storage tank. The exact cause of the primary explosion was never determined, but probably it was due to the presence of burning coal in the second pulverizer. The coal left in the pulverizer may have been overheated in the drier and have taken fire as it lay in the pulverizer; or the high sulfur content of this residual coal may have caused what is known as "spontaneous combustion." A sample subsequently taken showed that the coal left in the pulverizer contained 12.9 per cent sulfur. Evidently the air classifier and the centrifugal action of the pulverizer had a concentrating effect, as a result of which the heavy iron sulfide in the coal accumulated in the pulverizer.

Had each pulverizing unit been equipped with separate cyclone collectors over the storage tank, according to the usual practice, the second explosion might have been avoided entirely; at least its effects would have been minimized.

The original driers, which were of the shaft type, in which a descending column of coal was dried by a stream of hot gas passing through it,

were found to be unsatisfactory for either fine coal or very wet coal. Recently these driers were removed and a double-shell Ruggles-Coles drier was installed, which has made it possible to use bituminous coal of any good grade regardless of its physical condition.

Before September, 1928, Pennsylvania bituminous coal was used, but at that time tests showed that Nova Scotia coal gave equally good furnace results and since then Nova Scotia coal has been used exclusively. A typical analysis of this coal on a dry basis is as follows: volatile matter, 34.9 per cent; fixed carbon, 57.1; ash, 8.4; sulfur, 3.5; B.t.u. per pound, 14,100. The raw coal is sent either to the storage silos, which have a capacity of 4000 tons, or to the storage bin ahead of the drier. The coal is fed to the drier by a screw conveyor and discharged from the drier into a Redler elevator, which carries the coal to the dry-coal storage bin above the pulverizer feeders. The raw coal contains about 4 per cent moisture and is dried to 1 per cent moisture by using some of the waste gases from the reverberatory furnaces brought to the drier through an insulated steel pipe leading from the header flue of the reverberatory furnace. The drier fan is driven by a variable-speed motor, so that the quantity of gas used for drying can be varied to take care of changes in the quantity or the moisture content of the coal being dried. Under normal conditions the gases enter the driers at about 1000° F. and leave at about 130° F. A furnace has been provided at the drier so that if waste gases are not available either lump coal or pulverized coal can be burned to provide heat for drying.

The dried coal is fed to the pulverizers by two Hardinge constant-weight feeders driven by variable-speed motors controlled by rheostats located at the reverberatory furnaces, so that the quantity of coal going to the furnaces can be varied at will by the furnace operators.

The pulverizers are of the Holbeck-Bonnot type and are equipped with classifiers set directly over them. Classifying air is admitted at the bottom of the classifier and the air and coal mixture is drawn off at the top by the fan that blows the coal to the furnaces. The fineness of the coal depends on the quantity of air passing through the classifier, and control over the degree of fineness to which the coal is pulverized is obtained by changing the size of the opening in a plate inserted in the discharge side of the fan. This changes the volume of air passing through the classifier and thus the fineness of the coal produced. The coal pipe leading from the fan to the furnace branches into two pipes just outside the coal plant, and each branch is again split at the furnace, thus supplying coal to each of the four burners. Deflectors at each Y regulate the quantity of coal that is being sent through each branch.

The fan discharge of the third or spare unit is connected to both the other fan discharges, so that the spare unit can be used on either furnace. Positive shut-off valves must be used to prevent leakage of pulverized

coal into the unit that is not being used. Butterfly valves were used originally but caused trouble and much lost time when changing units because the pulverized coal formed a hard deposit on the edges of the valves and prevented them from seating properly. Finally a fast, positive-acting shut-off valve was devised which has done away with these difficulties. This valve (Fig. 8) was designed by J. Parry, coal-plant

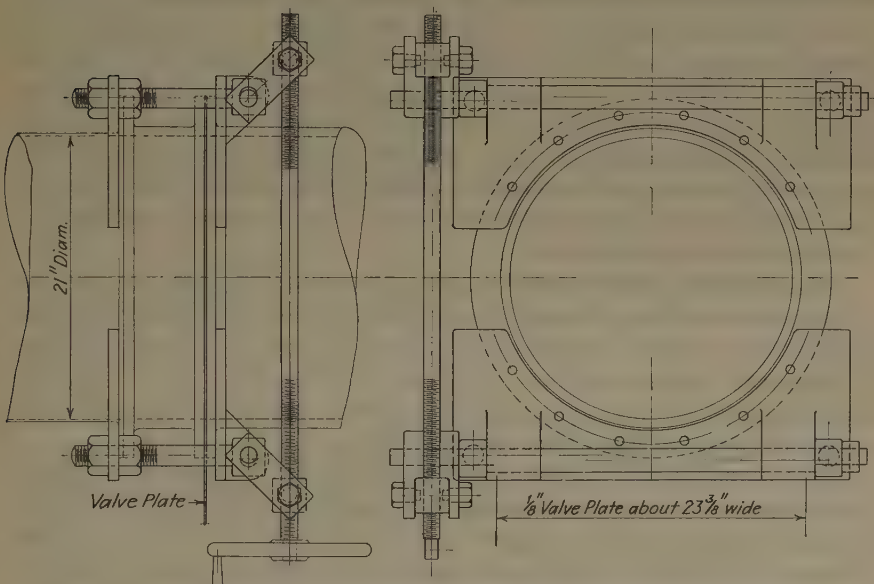


FIG. 8.—JACK VALVE DESIGNED BY J. PARRY TO SHUT OFF PULVERIZED COAL.

foreman, in conjunction with the engineering department. It consists of a lever and threaded screw arrangement by means of which the two flanges of a joint in the pipe may be separated and a plate inserted to close off the pipe.

Coal is supplied to the roaster burners by means of a circulating system. Pulverized coal and air may be "bled" off either or both reverberatory-furnace lines into the inlet of a fan that circulates the coal through the roaster burner system, the unused coal being returned to the fan inlet. This arrangement is not entirely satisfactory because it causes variations in the furnace fires and gives poor control over the roaster fires, but the quantity of coal used in the roasters is not now sufficient to justify the installation of a separate unit.

Cottrell and Stacks

Originally there was only one stack and one Cottrell plant consisting of six units in parallel, four of which were used for roaster gases and two for converter gases. When two more converters were added, it was neces-

sary to build a new stack for the converter gases alone in order to provide sufficient draft, and a two-unit Cottrell plant was also built, the original six-unit treater being used for roaster gases only.

Both treaters are of the same general design, known as the "rod curtain" type. Each unit consists of three sections in series, through which the gases travel horizontally. In the roaster treater there are 16 collecting electrodes in each section, the electrodes consisting of vertical pipe "curtains" suspended parallel to the direction of the gas flow and spaced 8 in. apart. The pipe curtains are 18 ft. high and 7 ft. 4 in. long and are made up of ⅛-in. pipes spaced 1½ in. apart. The discharge electrodes are ⅛-in. square twisted steel rods suspended at 8-in. intervals between the collecting electrodes. The two units of the converter treater are also in parallel and are similar to those of the roaster treater except that there are 19 collecting electrodes instead of 16 in each section, the units being 20 per cent wider.

High-tension unidirectional current is supplied to the treaters by mechanical rectifiers. There is a separate 25-kva. transformer for each rectifier, the primary voltage being 550 volts; taps are provided to give a secondary voltage of from 55,000 to 75,000 volts. There are two rectifiers for each unit, one supplying one section and the other two sections; each rectifier uses about 10 kilowatts.

The discharge and collecting electrodes hang on separate insulated suspensions and pneumatic hammers are used for rapping the electrodes to shake off the dust. The dampers in the outlets of the units are of the balanced multivane type, motor-operated. Rapping and damper-control arrangements enable a single attendant operating push buttons on a central control board to cut a unit out of service by closing the damper, rap the electrodes to clean off the dust, and open the damper again. In the roaster treater the rectifiers and the control board are near the units, while the rectifiers and control board of the converter treater are in the power house about 200 ft. from the treater, in order to enable the power-house attendant to do the necessary operating.

The recovered dust falls into V-shaped hoppers, one under each unit, and is drawn off periodically by screw conveyors into charge cars in which it is taken to the reverberatory furnaces.

The roaster treater is handling a volume of approximately 450,000 cu. ft. per min. and gives a recovery of 95 per cent of the dust in the gases. The converter treater was built to handle 200,000 cu. ft. per min. with a recovery of 94 per cent but during the last year it has handled 350,000 to 400,000 with a dust recovery of from 70 to 65 per cent.

The two stacks are identical in size and construction; they are 422 ft. 6 in. high and the outside dimensions are 31 ft. at the base and 20 ft. at the top; the inside dimensions are 23 ft. at the base and 18 ft. at the top. They are built of reinforced concrete with a 4-in. brick lining supported in

30-ft. sections on projections or shelves in the concrete. Each stack is provided with a ladder, a gas-sampling platform and lightning rods.

Accessory Buildings and Equipment

Power House and Supply

Power is brought from the Quinze plant on the Northern Quebec Power Co., a distance of about 60 miles. The 25-cycle current is received at 110,000 volts and is reduced to 12,000 volts in the power company's substation, which serves as a distributing point for the surrounding district. The current is received by the smelter at 12,000 volts and is distributed to several separate transformer stations at convenient points in the plant and reduced to 550 volts for use in the motors. Other transformers, where necessary, reduce the 550-volt current to 110 volts for lighting purposes.

The power house is a brick building 65 by 182 ft., west of and parallel to the converter building. It contains the following equipment:

1. Five Bellis and Morcom air compressors, capacity 2500 cu. ft. per min. at 100 lb. per sq. in., for supplying compressed air for use in the mine. Each is direct-connected to a 500-hp. synchronous motor.

2. Two Ingersoll-Rand air compressors, capacity 500 cu. ft. per min. at 100 lb. per sq. in., for supplying compressed air for use in the shops, and the smelter. Each is belt driven by a 100-hp. synchronous motor.

3. Two Ingersoll-Rand blowers, capacity 21,750 cu. ft. per min. at 15 lb. per sq. in., for supplying air to the converters. Each blower is direct-connected to a 1300-hp. synchronous motor.

4. One Bellis and Morcom blower, capacity 15,000 cu. ft. per min. at 15 lb. per sq. in., added to supply air for the third converter. It is direct-connected to an 1100-hp. synchronous motor.

5. One Brown-Boveri centrifugal blower, capacity 35,000 cu. ft. per min. at 15 lb. per sq. in., added to supply air for the fourth converter and as a spare. It is driven through a speed increaser by a 2800-hp. synchronous motor.

6. Two motor-generator sets for supplying direct current for synchronous-motor excitation and for supplying the cranes and the storage-battery charging panels. Each set consists of a 575-hp. induction motor direct-connected to a 60-kw. 125-volt d.c. generator and to a 430-kw. 250-volt d.c. generator.

7. One motor-generator set installed for the trolley system. It consists of a 600-hp. induction motor direct-connected to a 400-kw. 600-volt d.c. generator.

8. A large set of storage batteries for emergency converter service.

Miscellaneous Buildings

The general office is close to the plant buildings. In it is an automatic dial telephone system which interconnects all parts of the plant, the staff

residences and the local exchange. The laboratory, which is adjacent to the general office, contains sufficient equipment for all the requirements of mine, smelter and concentrator analytical and assay work.

The machine shop and plate shop are equipped to handle almost any repair problem that may come up. Some of the principal items of equipment are:

Open side planer 42 by 42 in. by 10 ft. 0 in.	Punch and shear
62-in. boring machine	1100-lb. air hammer
Lathe, 14 in. by 8 ft. 0 in.	Plate-bending roll, 72 by 1 in.
Lathe, 54 in. by 31 ft. 0 in.	Hydraulic press, 250 ton
Lathe, 20 in. by 14 ft. 0 in.	Lincoln electric welding machine
Shaper, 24 by 26 in.	Flanger
Milling machine	2 10-ton overhead cranes
Keyseater	2 power hacksaws
2 pipe-threading machines	Universal grinder
4 drill presses	Floor grinder

Two large warehouses are accessible by teams and standard railway cars and are equipped with a hoist and crawl for handling heavy materials.

The smelter change building contains a first aid room, in addition to a washroom, shower baths and lockers.

Treated water from the town water supply is used for drinking and toilet water and for fire service. Raw water is used for the cooling of the compressors and blowing engines, for the anode-casting wheel and other uses at the furnaces.

All parts of the surface plant are interconnected with standard-gage railway track, of which there is about 5 miles. A 40-ton trolley locomotive is used for handling coal and copper cars and for general yard switching. Incoming or outgoing material may be weighed on a track scale of 125 tons capacity.

The buildings are practically all of steel framework, and those requiring protection from the cold in winter, such as Cottrell, power house, shops, change houses, laboratory, office and warehouses, have brick walls. Those in which heat is not important, or in which considerable heat is generated, have low brick walls at the ground level, with the sides above covered by Robertson corrugated protected metal. Roofs are generally flat and consist of gypsum slabs, covered with five-ply composition tar and gravel roofing. Conveyor bridges and track sheds have Robertson corrugated protected metal roofs.

Smelter Operation

The smelter flow sheet is shown in Fig. 9. With a bedding system it is usual to calculate the roaster charge that must be used to produce a desired reverberatory-furnace slag. This predetermined charge is obtained by mixing in the beds the correct proportions of the various ores and fluxes and is then fed to the roasters. At Noranda, where there is no

bedding system, this procedure cannot be followed and the analysis of the furnace slags is controlled by changing the quantities of smelting ore and fluxing ore being fed to the roasters. Ordinarily, the concentrates are used as fast as they are received because of the danger of their catching fire if stored too long. The judgment of the furnace foremen, supplemented by samples taken at regular intervals and analyzed by rapid methods, must be relied upon to keep the slags as nearly correct as possible.

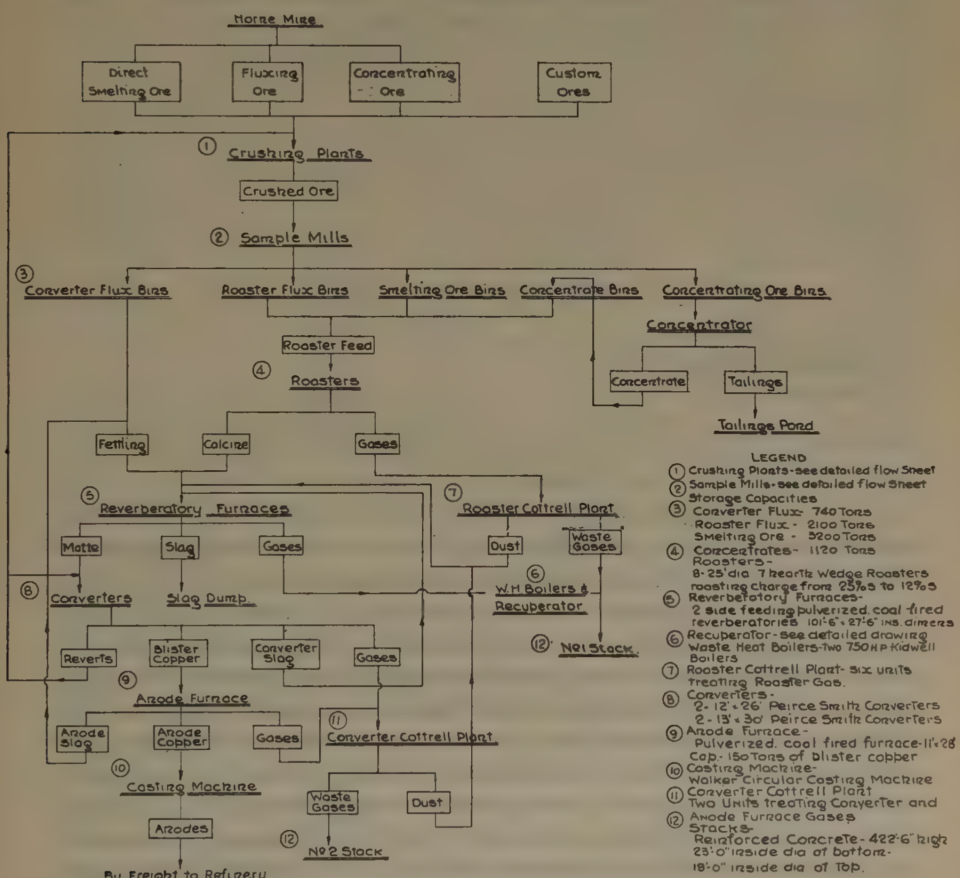


FIG. 9.—SMELTER FLOW SHEET.

It is obvious that this arbitrary method of control results in considerable variation in the analyses of the reverberatory-furnace slags made from hour to hour, but it is a limitation imposed by the absence of a bedding system. That any method of calculating the roaster fed is impossible will be apparent from the fact that if the storage bins are nearly empty material may be received and smelted before its analysis is known.

As in all smelters, clean slags are of vital importance for efficient operation. This is particularly true at Noranda, where, on account of the low copper content of the charge, the slag fall is high and a slight increase in the copper content of the slag means a considerable decrease in recovery. A complete analysis of the reverberatory-furnace slag for January to June, inclusive, 1933, is shown in Table 1. The alumina content is

TABLE 1.—*Analyses of Ores and Intermediate Products*
JAN. 1 TO JUNE 30, 1933

Material	Percentages						
	Cu	SiO_2	Fe	S	Al_2O_3	CaO	MgO
Smelting ore	3.28	16.1	39.6	26.9			
Fluxing ore	0.75	56.3	12.8	9.2			
Concentrates	10.81	4.2	41.8	38.2			
Roaster feed	3.82	24.2	33.6	24.3			
Calcine	4.05	26.3	36.2	11.8			
Roaster Cottrell dust	7.41	15.8	36.8	7.2			
Converter Cottrell dust	27.94	24.8	16.6	9.2			
Converter balloon dust	47.94	8.0	20.0	16.2			
Converter reverts	6.57	24.5	42.5	4.8			
Matte	19.3		48.3	22.5			
Reverberatory slag	0.27	37.9	36.4	1.1	8.5	1.7	1.9
Converter slag	4.40	25.0	47.7		4.8	0.5	0.3

fairly constant, usually between 8 and 9 per cent, and the silica content must be from 37 to 38 per cent with a silica degree of 1.34; or from 1 to 2 per cent higher than the iron content, in order to give sufficiently clean slags.

Originally it was intended that limerock should be used as a flux, but this is justified only when it makes cleaner slags, which more than compensate for the increased costs and losses occasioned by its use. With the basic materials smelted at Noranda, the addition of limerock would increase the quantity of siliceous flux required, thus decreasing the quantity of sulfide ore and concentrates that could be smelted. The slags being made are sufficiently clean to make the use of limerock undesirable.

The smelting ore is a massive sulfide ore containing chalcopyrite, pyrite and pyrrhotite. The mineral composition is by no means constant, the relative amounts of the various sulfides varying considerably in different orebodies and in different parts of the same orebody. The concentrates consist almost entirely of chalcopyrite and pyrite. The fluxing ore is mined in the Horne mine and consists of an acid flow rock mineralized with sulfides. It carries sufficient metal values to be classed as a revenue-bearing material.

In order to give more uniform mixing of the roaster charge than would be possible in the roasters alone, some fluxing ore is added to the smelting

ore by feeding it to the jaw crusher at the 125-ft. level when smelting ore is being crushed. The silica content of the smelting ore is fairly constant and it is possible, by this means, to obtain a uniform mixture of smelting ore and fluxing ore which contains from 20 to 28 per cent SiO_2. The quantity of fluxing ore used in this way is about 25 per cent of the weight of the smelting ore. Additional fluxing ore is crushed and stored separately, of course, and fed to the roasters in the quantity required to make the desired furnace slags. Table 2 shows the quantities of the various materials that made up the roaster feed for January to June inclusive, 1933; also the quantity of calcine produced and the dust recovered in the roaster Cottrell plant in the same period. In Table 1 are given the composite analyses of the ores and concentrates smelted and the analyses of various intermediate products for the same period.

TABLE 2.—*Roaster Feed, Jan.* 1 *to June* 30, 1933

Material	Dry Tons	Per Cent
Smelting ore	274,032	64.9
Fluxing ore	80,435	19.1
Concentrates	67,168	15.9
Custom ores	34	
Reverts	519	0.1
Total feed to roasters	422,188	100.0
Calcine produced	384,459	91.2
Roaster Cottrell dust produced	17,705	4.2

Under present conditions, the matte made in the reverberatory furnace contains from 18 to 24 per cent copper. Fixing the grade of matte to be made is not merely a roasting problem but one that involves the consideration of other factors such as furnace capacity, fuel ratio, relative reverberatory-furnace and converting costs, and plant tonnage as a whole.

In a reverberatory-furnace smelting operation the grade of the matte produced depends on the copper-sulfur ratio of the roaster feed and on the amount of sulfur eliminated in the roasters and reverberatory furnaces. At Noranda, the copper content of the roaster feed is comparatively low, varying from 3.5 to 5 per cent copper, and to produce high matte it would be necessary to secure high sulfur elimination in the roasters and reverberatory furnaces. The elimination of sulfur in a reverberatory furnace depends on reactions between oxides, sulfides and sulfates in the charge and on the oxidation of the sulfides of the charge by oxygen in the furnace gases. The latter can be increased by using a great excess of combustion air and high draft, but this does not give good firing conditions. The function of a reverberatory furnace is to smelt a maximum tonnage of

solid charge with the best possible fuel ratio. On the accomplishment of this depends the ability of the smelter to treat a large tonnage of ore at low unit cost and the elimination of sulfur in the reverberatory furnace, although desirable, is only incidental.

Sulfur elimination in the roasters is a more complex problem, governed by variable factors such as the mineral composition, the size and the sulfur content of the roaster feed. Decrepitation, which favors more complete roasting by presenting a greater surface for oxidation, does not occur with the Horne sulfide ore, probably because of the quantity of pyrrhotite present. The roasting of the ores and concentrates results in the formation of oxides and sulfates of iron and copper. The iron oxides produced are Fe_2O_3 and Fe_3O_4 (magnetite), and with the exception of a part of the Fe_3O_4 which goes out with the slag, these oxides must be reduced to FeO in the reverberatory furnace before the iron can be fluxed and removed as slag. Two of the more important reactions in the reduction of the iron oxides are as follows:

$$3Fe_2O_3 + FeS \rightarrow 7FeO + SO_2$$
$$3Fe_3O_4 + FeS \rightarrow 10FeO + SO_2$$

The resulting FeO unites with the silica of the charge to form slag while the SO_2 escapes with the furnace gases. It follows that the speed with which the iron oxides can be reduced to FeO depends on the quantity of iron sulfide present in excess of that required to satisfy the equations. For this reason, a charge low in sulfur is more refractory than one high in sulfur.

Table 3 shows the quantity of magnetite in the calcines, converter slag, and reverberatory furnace slag for the month of May, 1933. About 88 per cent of the magnetite in the furnace charge is reduced in the furnaces. As much magnetite as possible should be reduced in the furnace because excessive magnetite in the slag causes high slag losses and if magnetite settles out in the furnace bottom it causes trouble in tapping the matte.

Another consideration in controlling the degree of roasting is that the calcine from a "dead" roast is not as hot as that from a "green" roast and it is well known that the temperature of the furnace feed has a great effect on the tonnage that can be smelted in a reverberatory furnace. The grade of the matte produced has a further effect on the smelter tonnage. The total quantity of flux used in the roasters and converters to produce a required slag from a given tonnage of smelting ore and concentrates of fixed analysis is constant. The quantity of converter flux required (assuming a constant converter-slag analysis) depends on the quantity of matte produced. If the grade of the matte is lowered and the quantity thus increased more of the fluxing ore must be used as converter flux, which reduces the quantity of flux to be fed to the roasters, and thus reduces the quantity of calcine that would be produced in making up a

smelting mixture from the given tonnage of smelting ore and concentrates. But since the tonnage of calcine that can be smelted in the reverberatory furnaces remains the same, it follows that increasing the quantity of converter flux used by lowering the grade of the matte will increase the total tonnage smelted by the additional quantity of converter flux required by the lower grade of matte.

TABLE 3.—*Quantity of Magnetite for One Month*[a]

Material		Fe_3O_4			
	Tons	Per Cent	Tons	Input, Per Cent	Reduced in Reverbs, Per Cent
Calcine	64,703	10.4	6,729	47	
Converter slag	42,550	17.2	7,319	53	
Reverberatory slag	70,330	2.3	1,618	12	88

[a] This table was compiled from actual tonnages and analyses for the month of May, 1933.

For these reasons the roast is so controlled that the quantity of matte is sufficient to keep all the available converters operating at full capacity. Ordinarily, four converters are available for about two-thirds of the time, one being kept under repairs during the rest of the time. Up to the present there has been only enough converter air available for three converters, so that when four converters are tapped into, only three are blown at a time. With the new centrifugal blower recently installed there is converter air capacity sufficient for four converters but as yet it has not been used.

As has been mentioned, the size of the roaster feed is an important factor in the control of smelting operations. The finer the charge, the more easily it can be roasted and the hotter the calcines will be for the same sulfur elimination. Up to a certain point, the finer the charge to the furnaces, the more easily it can be smelted, partly because of the more intimate mixture of the materials of the charge and partly because of the lower angle of repose in the furnace, which exposes a greater surface of charge to the heat of the flame. If the charge is too fine, however, it slides too far into the furnace, filling up the bath and reducing the area available for combustion. For this reason, the roaster feed is crushed as fine as possible without producing excessive sliding in the reverberatory furnaces.

ROASTER OPERATION

The roasters as designed were rated at 150 tons of feed per roaster day. After it was found possible to smelt 1000 tons a day in one reverberatory furnace, it became evident that when the second unit was started it

would be necessary to almost double the tonnage per roaster day, since there were only four roasters for each reverberatory furnace. Accordingly, considerable experimental work was done to find the maximum tonnage of feed that the roasters were capable of handling with satisfactory sulfur elimination.

The original speed of rotation of the roaster shaft was one revolution in 80 seconds. This was gradually increased to the maximum possible, which was found to be one revolution in 55 seconds; higher speeds than this produced excessive vibration and wear in the moving parts. At this speed the capacity of the roasters is about 325 tons of feed per roaster day, reducing the sulfur content from 24 per cent in the feed to 11 or 12 per cent in the calcine.

Increasing the feed to the roasters brought some troubles with it. The temperatures throughout the roasters were increased, a maximum of 1400° to 1500° F. being reached on the third and fourth hearths, and these high temperatures together with the increased gas volume caused the fine particles carried up by the gases to sinter on the underside of the second and fourth hearths, above where the gases came up through the outside drop-holes on the third and fifth hearths. Chemical analysis showed that this material was mostly oxidized pyrite fines which "flashed" in the blast of gas and collected as a fused mass on the underside of the second hearth. It abraded the rabble arms and was a source of constant trouble until a mechanical means was found for removing it, by devising a plough to be attached to the top of a rabble arm on the third hearth.

An attempt was made to reduce the gas velocity through the drop-holes by bleeding a part of the gases from the fourth hearth directly into the roaster uptake. This experiment was unsuccessful, because it lowered the temperature of the top hearths and therefore reduced the efficiency of the roaster. It is impracticable to increase the size of the outside drop-holes without completely rebuilding the hearths. The area of the opening on the center drop hearths was increased by removing two courses of the hearth brick.

Other sources of trouble have been the breaking of the cast-iron lute rings that form the seal between the outside drop hearths and the shaft, and the losing of the brick on the central shaft. Lute-ring trouble has been eliminated by minor changes in design; the column brickwork has been improved by supporting the sections of brickwork at every hearth instead of at every second hearth.

Various designs and materials have been used for the rabble blades, which are carried on hangers that slide on the roaster arms. As originally designed the rabble holders were the same on both the out and the in hearths, while the rabbles were different. It has been found to be more convenient to have different holders for the out and in arms on the differ-

ent hearths and to use interchangeable blades. Various materials have been used for blades, the results showing that white iron is most economical for the dry and first-hearth blades, heat-resisting chrome-nickel alloys for the hot hearths, and cast iron for the two bottom hearths.

Ordinarily a roaster must be shut down about once a year for general overhaul. Considerable wear on rabble blades can be saved by ploughing up the hearths and rebedding them with siliceous ore, at each major roaster repair.

The charge is not self-roasting, from 5 to 10 lb. of pulverized coal being required per ton of calcine. There are two pulverized-coal burners on each roaster, one on the third hearth and one on the fifth. The method of supplying the coal to the burners was described earlier in this paper.

The roaster gases leave the roaster at about 1000° F. and contain approximately 5 per cent SO_2 by volume. About 4.5 per cent by weight of the roaster charge is recovered as dust in the roaster Cottrell plant.

Roaster draft varies from 0.20 to 0.40 in. water gage. It is measured at the inlet header of the Cottrell and is controlled by adjusting the outlet dampers of the treaters.

The temperature of the calcine as it falls into the hoppers below the roasters varies from 1100° to 1250° F.

Operation of Converters and Anode Department

The only unusual feature of the converter operation is the treatment of large quantities of low-grade matte to make a comparatively small production of copper. Four converters, only three blowing at a time, are used, except for periods when a converter is being relined. Because of the shortage of blower capacity, air pressure at the converters recently has been only about 9 lb. at the converters, but with the new blower it is expected that 12 lb. will be obtained. The 12 by 26-ft. converters have forty 1½-in. tuyeres and the 13 by 30-ft. converters have forty-four 1½-in. tuyeres.

Practically no converter reverts are now returned to the roasters or reverberatory furnaces for resmelting. All floor cleanings, ladles skulls and converter-slag launder cleanings are treated in the converters. The converter slags contain about 25 per cent SiO_2.

The copper is blown to blister copper in the converters and is then transferred to the anode furnace. Ordinarily, three converter charges are required to fill the anode furnace. The blister copper contains about 98.5 per cent copper. The copper is usually made on the afternoon and night shifts and the anode charge is refined to 99.4 per cent copper and cast into anodes on the day shift.

Reverberatory Furnace Practice

The original furnace construction is shown in Fig. 10. Two of the unusual features were the bridge wall about 12 ft. from the firing end of

the furnace and the hot-air ducts which led up through the furnace bottom between the firing wall and the bridge wall, and served to introduce the air from the preheater into the furnace. This construction gave trouble and had to be changed to a more orthodox design.

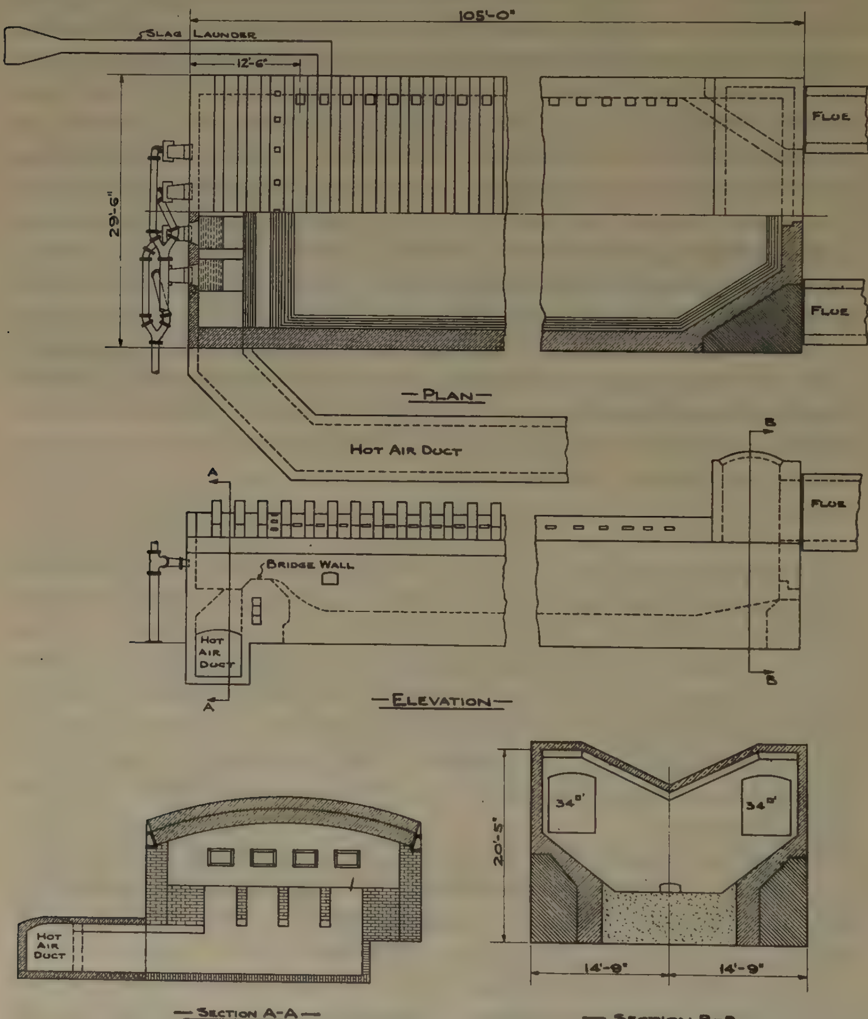


Fig. 10.—Original construction of reverberatory furnace.

The bridge wall was exposed on both sides to the heat of the coal flame and started to burn out soon after the furnace was "blown in" in December, 1927. After six days most of the bridge wall was gone and the hot-air passage under the furnace was partly filled with slag which had overflowed from the furnace. At the suggestion of J. Howard, the general foreman, the duct from the preheater was bricked off and the air

passages under the furnace were completely filled with molten converter slag. The bridge wall was allowed to burn out completely, and additional charging holes were provided for introducing the charge along the side walls between the firing wall and the original position of the bridge wall. The converter-slag launder was too long and flat, and its position on the side wall prevented the use of some of the added charging holes. It was moved so that the converter slag could be poured into a shorter, steeper launder through the firing wall of the furnace. The use of preheated air was temporarily abandoned while these changes were being made, but afterwards pipes were provided for bringing the hot air from the duct alongside the furnace into the bottom half of the furnace burners.

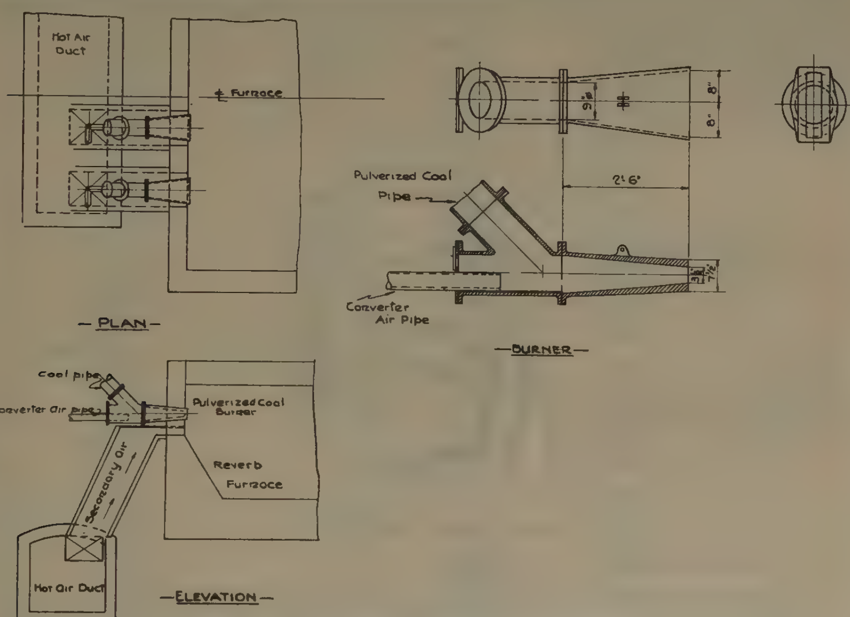


FIG. 11.—PRESENT BURNER ARRANGEMENT.

These pipes were comparatively small and some time afterwards when the furnace tonnage was being increased, and it was necessary to burn more coal, the burner design was changed and these pipes were removed, ducts for the hot air being provided under the burners, as shown in Fig. 11.

After these changes had been made the quantity of solid charge smelted per furnace day was about 700 tons. The first big improvement was made by J. Howard, the general foreman, by introducing air from the converter blowers into the furnace burners. Fig. 11, which shows the present burner arrangement, illustrates how the converter air was introduced, except that the first pipes used were only 2 in. in diameter, as compared with the 4-in. pipes now used. Providing this additional amount of air made it possible to burn more coal and gave a short,

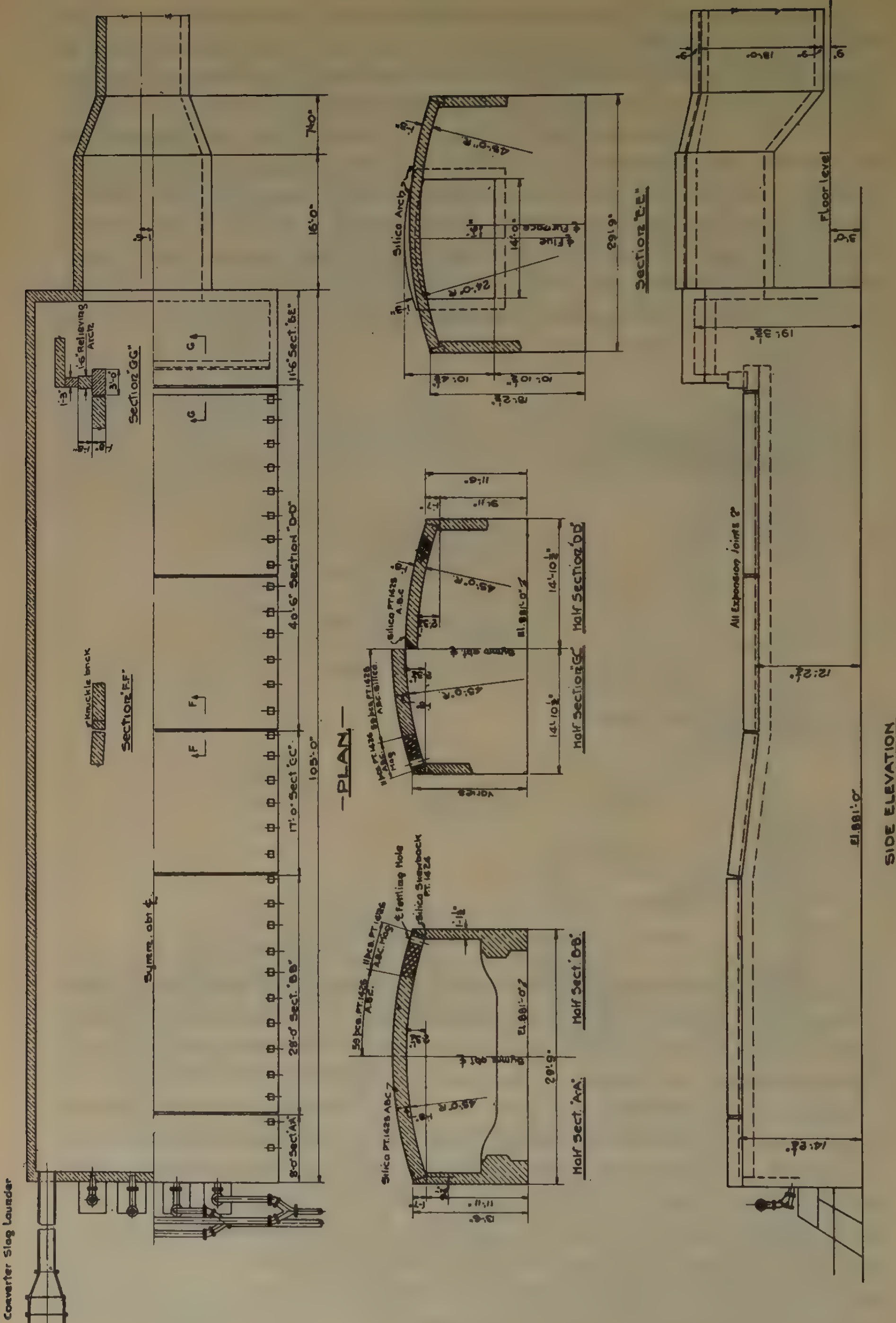


FIG. 12.—PRESENT FURNACE CONSTRUCTION.

intensely hot flame, which increased the furnace capacity from 700 to 900 tons of solid charge per furnace day.

Increased outlet-flue area was required to allow more coal to be burned therefore this area was increased from 68 sq. ft. to 132 sq. ft. The uptake arch construction shown in Fig. 10 gave trouble and was changed to a straight-line arch. These changes together with the changes at the firing end already mentioned were incorporated in the design of No. 2 furnace, which was only partly finished when the smelter was started.

The present furnace construction is shown in Fig. 12. It will be noticed that the double outlet has been changed to a single outlet. This change was made early in 1933 when a header flue for the two furnaces was built in order to supply furnace gases to the waste-heat boilers.

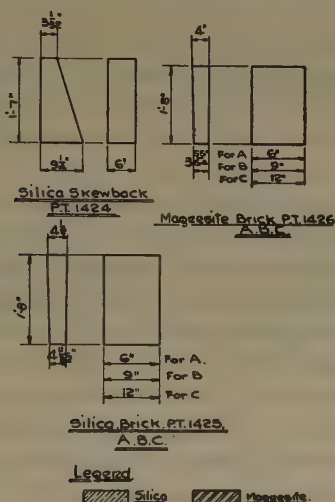


FIG. 12.—(*Continued*).

The area of each present furnace outlet is 153 sq. ft. The reasons for and the effects of the various other improvements in the furnace construction will be found in the following discussion.

Experience has shown that the four most important requirements for good firing conditions in a reverberatory furnace are: (1) dry finely pulverized coal; (2) absolute control over the quantities of coal and combustion air; (3) complete combustion of the coal without the use of an undue excess of air; and (4) obtaining a short, hot flame by intimate mixing of the coal and all the combustion air as soon as possible in the furnace.

The fineness of the pulverized coal is from 80 to 85 per cent minus 200 mesh. Still finer coal would give better results, but this is the best that can be done with the present equipment. Proper combustion depends on having positive regulation over both the coal and the combustion air. As has been mentioned, the quantity of coal can be regulated at will by

the furnace operator. The combustion air is supplied as primary air with the coal, as secondary air under the burners, and as converter air reduced to 5 lb. in the burners. The quantity of primary air is constant, the quantity of secondary air is small and subject to control, and the converter air may be controlled by valves on each of the pipes leading to the burners.

Our experience has shown that the best firing conditions are obtained with from 0.5 to 1.0 per cent oxygen in the furnace gases, sampled at a point 75 ft. from the firing wall of the furnace. With a lower oxygen content, carbon monoxide is usually present, indicating a long flame and incomplete combustion in the smelting zone; the excess of air indicated by a higher oxygen content decreases the flame temperature and thus the rate of smelting.

A water-cooled sample tube is used for taking the gas samples, and the analysis is made with an Orsat apparatus. The analysis of a typical sample is as follows: $CO_2 + SO_2$, 18.2 per cent; O_2, 0.8; CO, 0.0. The gases usually contain from 1.0 to 1.5 per cent SO_2, depending on the nature of the charge. The furnace draft is measured at the gas-sampling hole, and is kept between 0.08 and 0.10 in. water gage by regulating the dampers in the bypasses at either end of the furnace header flue.

Since 1929 the capacity of the reverberatory furnaces has been increased from 900 to 1267 tons of solid charge per furnace day in December, 1933. The various factors that have resulted in this increase, in the approximate order of their development, are given below:

1. The use of more air from the converter blowers.
2. Widening the furnace by decreasing the thickness of the side walls and increasing the effective width of the furnace by moving the drop-holes as close to the side walls as possible.
3. Increasing the cross-sectional area of the combustion chamber by raising the furnace arch.
4. The use of hot air from the preheater to furnish hot primary air.
5. Operating the roasters to give calcines as hot as possible.
6. Using a charge as fine as possible.

Quick, complete combustion of the coal requires an intimate mixing of the coal and the combustion air as soon as the mixture is introduced into the furnace. By using converter air, it is possible to introduce a large proportion of the combustion air into the burner, with the result that a short, intensely hot flame is produced. Each increase in the size of the converter air pipe in the burners has been marked by an increase in furnace tonnage. The pipes now being used are 4 in. in diameter and it is estimated that they supply from 45 to 50 per cent of the combustion air. About 35 per cent of the combustion air is introduced as primary air; this cannot be increased because the mixture would then be liable to catch fire or explode.

The quantity of fuel that can be burned efficiently depends on the flue area of the furnace and the effective cross-sectional area of the combustion chamber. Each furnace now has an outlet-flue area of 153 sq. ft. and the combustion area in the smelting zone has been made as large as possible by decreasing the thickness of the furnace walls from 27 to 13½ in. and by raising the arch of the furnace in the smelting zone 24 in. As shown in Fig. 12, the raised section of the arch extends from the firing wall 50 ft. towards the front of the furnace, and a sloping section 11½ ft. long connects the raised section with the rest of the furnace arch. Various lengths of both the raised and the sloping sections have been tried and it has been found that the construction shown gives the most satisfactory furnace tonnage together with longer life for the sloping section of the arch.

Moving the drop-holes close to the side walls has increased the available combustion area and minimized the burning out of the side walls, because more protection is provided to the side walls by the change.

The use of hot primary air has been discussed under the operation of the preheater.

The roasters are operated to make the calcine as hot as possible without producing an excessive quantity of matte. Heat losses from calcine are minimized by emptying the roaster hoppers every half hour, each side of the furnace being charged alternately, and by storing no calcine in the furnace feed bins. Roaster and furnace operation must be coordinated so that the quantity of calcine being produced takes care of furnace requirements without excess or shortage. Roaster shutdowns for excess calcine are held to the absolute minimum.

As has been mentioned, the fineness of the roaster feed is controlled so that it is as fine as possible without producing excessive sliding in the furnace. If the charge slides so much that it meets in the center, the flow of slag and matte to the skimming end of the furnace is interrupted and they overflow at the burner end while at the same time there is little or no slag at the skimming end. If this condition occurs charging must be discontinued and more coal put on the center burners to smelt the material in the center of the furnace.

The smelting of roaster Cottrell dust has given more trouble as the quantity recovered has increased, owing to more concentrates on the charge and higher roaster tonnages. This material from its nature slides badly in the furnace. Present practice is to charge small quantities of dust at frequent intervals and to confine the charging to about 20 ft. on one side of the furnace, thus localizing the effect of sliding.

In Table 4 are given the tonnages and fuel ratios for the reverberatory furnaces by years from 1928 to 1933. The fuel ratio is the percentage ratio of the coal burned in the reverberatory furnaces to the dry tons of solid charge, exclusive of the liquid converter slag. The weight of the

TABLE 4.—*Reverberatory Tonnages and Fuel Ratios*

Period	Calcine	Siliceous Ores Direct	Other Ores Direct	Roaster Cottrell Dust	Converter Dust		Reverts	Total Solid Charge	Liquid Converter Slag	Total Solid and Liquid Charge	Coal Burned in Reverbs, Tons	Coal per Ton Solid Charge, Per Cent	Slag Produced, Tons	Matte Produced, Tons	Furnace Days	Solid Charge per Furnace Day, Tons	Lost Time for Major Arch Repairs, Furnace Days
					Cottrell	Balloon											
1928	235,316	3,378		4,684		968		244,346	77,480	321,826	31,696	13.00	223,915	86,825	351.9	694	
1929	334,407	5,764	1,787	9,421		1,735		353,114	151,060	504,174	43,784	12.40	358,456	161,984	405.2	871	
1930	596,765	8,363	3,337	20,003	1,836	811	2,216	633,346	223,430	856,776	78,535	12.40	579,570	239,556	705.4	898	19.14
1931	606,681	11,666	3,892	27,949	1,953	2,888	9,309	664,338	249,320	913,658	81,321	12.24	614,939	244,357	706.3	941	17.79
1932	707,725	15,388	1,368	31,498	2,050	2,233	7,630	767,892	382,060	1,149,952	85,933	11.19	747,140	322,380	714.4	1,075	8.93
1933	786,220	8,590	48	38,214	2,007	1,597	8,829	845,505	460,460	1,305,965	90,045	10.65	813,991	377,232	708.4	1,193	15.43

coal used is obtained from the railway net weight of the coal received and includes any coal burned in the drier furnace. The tonnage of solid charge smelted per furnace day has been increased from 694 tons in 1928 to 1193 in 1933. The lost time for major arch repairs was reduced by better arch construction from about 18 furnace days in 1930 and 1931 to about 9 days in 1932. The high lost time for the first half of 1933 was due to shutdowns for the new furnace flue construction. In 1928 and 1929, when only one furnace was operating, there was little or no lost time for arch repairs because the second furnace was started up before the operating furnace was shut down for repairs.

In Table 5 are given the total tonnages of ores and concentrates smelted, the quantities of coal burned in the roasters, reverberatory furnace and anode furnace, and the ratio of the total coal burned to the total tonnage smelted by years from 1928 to 1933. The highest monthly average tonnage so far obtained was 1259 tons of solid charge per furnace day on No. 1 furnace in March, 1933. In February, 1933, while one furnace was down for repairs and there was plenty of roaster and converter capacity available, the other furnace smelted 1437 tons of solid charge in 24 hours.

Table 6 shows the percentages of the various materials that made

up the solid charge to the reverberatory furnace for the six-month period January to June, inclusive, 1933. The only materials in the solid charge that can be classed as secondary products are the converter Cottrell and

TABLE 5.—*Ores Smelted and Fuel Consumed*

Date	Total Ores and Concentrates Smelted	Coal to Roasters, Tons	Coal to Reverbs, Tons	Coal to Anode Furnace, Tons	Total Coal Burned, Tons	Percentage of Total Coal Burned to Total Ores Smelted
1928	271,926	172	31,696		31,868	11.72
1929[a]	428,221	964	43,784		44,748	10.45
1930[b]	734,073	6,809	78,535		85,344	11.63
1931	765,544	7,123	81,321	1,602	90,046	11.76
1932	918,568	11,023	85,933	1,448	98,404	10.71
1933	1,010,630	6,029	90,045	1,516	97,590	9.66

[a] One furnace operating 12 months; one furnace 1½ months.
[b] Both furnaces operating 12 months.

TABLE 6.—*Distribution of Furnace Feed*

Material	Per Cent Composition of Solid Charge to Reverbs Jan. 1 to June 30, 1933 of Total Solid Charge
Calcine from smelting ore	59.8
Calcine from fluxing ore	17.6
Calcine from concentrates	14.7
Reverts to roasters	0.1
Total calcine	92.2
Fluxing ore for fettling	1.4
Roaster Cottrell dust	4.2
Converter Cottrell dust	0.3
Total Cottrell dust	4.5
Converter balloon flue dust	0.2
Converter reverts direct to furnace	1.7
Total solid charge[a]	100.0

[a] 97.7 per cent of solid charge is new revenue-bearing material.

balloon flue dust and the converter reverts. The total of these three is 2.3 per cent, so that 97.7 per cent of the furnace charge is new revenue-bearing material.

In 1929, when the furnace capacity was 900 tons of solid charge per furnace day, the average arch life between major repairs was about 90 days. Since that time, various improvements in arch construction have been made and at present the interval between major repairs is from 150 to 200 days when smelting 1250 tons of solid charge per furnace day.

The part of the furnace arch that burns out most frequently extends from a point 10 ft. from the firing wall to a point 50 ft. from the firing wall. All the experimental work on arch construction has been done on this part of the arch and the figures given for life and tonnage smelted for a given arch apply to this section. In 1929 silica brick was used throughout for arch construction. The standard brick size was 20 by 6 by 3 in., the 20 by 9 by 3 in. size being used to break the joints. One course of No. 2 wedge brick was used with four courses of straight brick to give the required curvature of the arch. The longer arch life now being obtained is due to the following changes:

1. Raising the arch at the firing end of the furnace and moving the drop-holes closer to the side walls.

2. The use of magnesite shoulders on the arch.

3. The development of the use of wedge brick 20 by 9 by $4\frac{1}{2}$ to $4\frac{11}{32}$ in., designed so that no straight brick are required.

The first step taken was to use the 20 by 9 by 3-in. brick, with 20 by 12 by 3-in. brick to break the joints, which reduced the number of joints and leakage of air through the arch; in addition, the arch could be laid more quickly. The second step was to use No. 1 wedge brick instead of No. 2 wedges; the No. 1 wedge has less taper than the No. 2 wedge and only two courses of straights were required for each course of wedges, consequently the arch more nearly approximated the true arc of a circle and stresses in the arch were minimized. A logical development from these two steps was to use the 20 by 9 by $4\frac{1}{2}$ to $4\frac{11}{32}$-in. brick as the standard silica brick, and the 20 by 9 by 4 to $3\frac{7}{8}$-in. brick as the standard magnesite brick. The greatest thicknesses the brick manufacturers consider advisable from a manufacturing standpoint are $4\frac{1}{2}$ in. for silica and 4 in. for magnesite brick. The size and weight of the bricks are about the maximum for convenient handling in the laying of the arch.

Methods of Arch Construction Used and Results Obtained

In 1929 the firing end of arches of both furnaces was raised 13 in., a sloping section $11\frac{1}{2}$ ft. long being built to join the raised section to the rest of the arch; this did not lengthen the life of the arch much, although it did increase the furnace capacity. The average life of five all-silica arches, during the period from January, 1930, to October, 1930, was 100 days, with an average tonnage per furnace day of about 900 tons of solid charge. In the period from October, 1930, to August, 1931, four arches were replaced using 42 in. of magnesite brick on each side and

silica brick in the center. The 20 by 6 by 3-in. series of straights and No. 2 wedges being used, burned magnesite bricks were used for the first two arches and unburned magnesite brick, for the last two. The average life of these four arches was 140 days, with an average tonnage of 900 tons of solid charge per furnace day. There was no difference between the arch life obtained with burned and unburned magnesite brick because in every case it was the silica brick that burned out first. Unburned magnesite bricks are now used exclusively because they are cheaper than the burned bricks.

When No. 1 furnace was repaired in August, 1931, the 20 by 9 by 3-in. bricks were used as the standard and No. 1 wedges instead of No. 2 wedges and the arch was raised another 5 in. at the firing end. This arch lasted 254 days, with an average tonnage of 1000 tons of solid charge per furnace day. The next arch on No. 1 furnace was exactly the same except that it was raised 5 in.; it lasted 210 days with an average tonnage of 1050 tons of solid charge per furnace day. This arch was burned out prematurely because one of the burners was tilted up too much.

In the meantime, two unsuccessful experimental arches had been tried on No. 2 furnace. The first consisted of the regular unburned magnesite brick and silica brick of another manufacturer. This arch lasted 116 days as compared with 140 days for similar arches using regular silica brick. Premature failure was due to inferior quality, many of the shapes being warped and the bricks not being of good refractory properties. The second arch was a 15-in. arch built completely of unburned magnesite brick. An arch of all basic brick is desirable because silica brick are fluxed by the basic dust of the charge, and the purpose of this experimental arch was to find whether magnesite brick were strong enough to be used for the entire span of the arch. The arch failed after 37 days, the brick crushing and spalling under the heavy load at high temperature.

An intermediate step between using the 3-in. straights and No. 1 wedges and the 4½-in. all wedge brick was to use the all wedge 20 by 9 by 3 to 2⅞-in. series. An arch built of this brick on No. 2 furnace in July, 1932, lasted 226 days, with an average tonnage of 1125 tons of solid charge per furnace day.

Since then the larger sizes of silica and magnesite brick have been used for all arch construction at the firing end of the furnace. Except for the experimental all-magnesite arch, 20-in. brick had been used exclusively; in May, 1933, a section of the arch on No. 1 furnace was built using 15-in. brick, but this lasted only 89 days with an average tonnage of 1134 tons of solid charge per furnace day. In view of this performance, 20-in. brick seems to be the minimum for use at the firing end of the furnace.

No figures are as yet available for the arch life that will be obtained by the use of the large all-wedge brick (20 by 9 by 4½ to 4$^{11}/_{32}$ in.). One

arch on No. 1 furnace burned out in 165 days but its period of service was interrupted and seriously shortened by a four-day shutdown for the new outlet and header flue construction work done early in 1933.

The authors believe that the maximum life obtainable will be from 180 to 200 days when smelting 1200 tons of solid charge per day. It is not possible to use wider magnesite shoulder and thus increase the arch life because the magnesite bricks will not carry a greater load than they do now with a 44-in. width of shoulder. The only way in which the advantages of magnesite brick for the entire arch span can be utilized is by some form of suspended arch.

Conclusion

In Table 7 are given the total tons of smelting ore, concentrates and fluxing ore smelted and the production of gold, silver and copper by years from 1928 to 1932, inclusive. Table 5 shows that the quantity of material smelted in 1933 was 1,010,000 tons of new metal-bearing material.

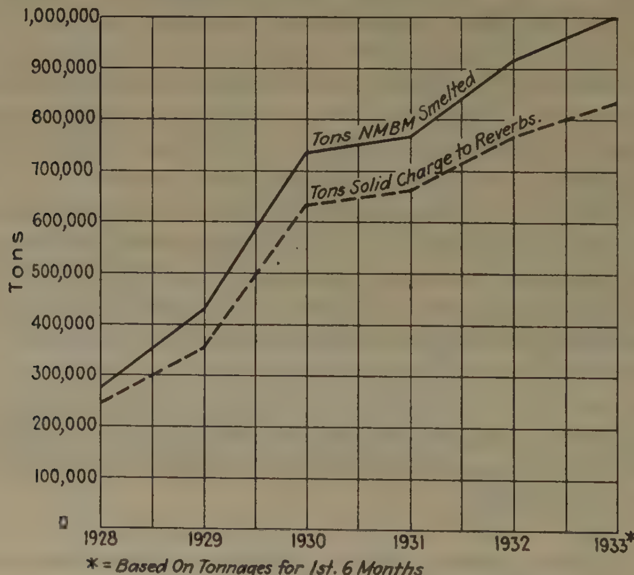


FIG. 13.—TONNAGES SMELTED FROM 1928 TO 1933.

In Fig. 13 are plotted the total tons and the tons of solid charge smelted by years for the period 1928 to 1933 inclusive. The present plant costs for material, labor, fuel and power are: material, 19.6 per cent of total cost; labor, 31.4; fuel, 41.1; power, 7.9. Fuel, labor and power make up over 80 per cent of the cost. In the period January, 1928, to June, 1933, there was no change in the scale of wage or in the price paid

for coal and power. The marked decrease in the cost per ton of new metal-bearing material smelted has been due almost entirely to the greater tonnage and increased efficiency.

TABLE 7.—*Tonnages Smelted and Metals Produced*

Year	New Metal-bearing Material Smelted, Tons	Production		
		Gold, Oz. per Ton	Silver, Oz. per Ton	Copper, Lb.
1928	271,926	52,949	186,277	33,065,261
1929	428,221	68,732	334,279	51,223,115
1930	734,072	117,393	691,920	75,509,373
1931	765,544	253,368	558,801	62,759,355
1932	918,567	341,350	619,597	63,013,485

ACKNOWLEDGMENTS

Acknowledgment is made to the management of Noranda Mines, Ltd., in giving the smelter operators a free hand in trying the necessary and sometimes unsuccessful experimental and development work, and for the privilege of publishing the data contained in this paper.

Acknowledgment is made also of the cooperation of the staff of the Mining Department of Noranda Mines, Ltd., through which smelter tonnage has not been limited by shortage of ore. The results obtained have been due, in no small part, to the ability, ingenuity and enthusiasm of the smelter operating personnel.

Smelting Operations at Roan Antelope Copper Mines, Limited

By Charles R. Wraith,* Luanshya, Northern Rhodesia, Africa

(New York Meeting, February, 1934)

The property of the Roan Antelope Copper Mines, Ltd., is situated at Luanshya, north central part of Northern Rhodesia, and is connected with the main line of the Rhodesia Railways, Ltd., by a 24-mile branch line from N'dola. Most of the material and supplies that are not available in Africa are shipped into the country via Beira, on the East Coast of Africa, a distance of 1473 miles from Luanshya and the nearest ocean port. The output of copper is shipped by rail to Beira and thence by boat to the market. Passenger travel is mostly via Cape Town, a distance of 2162 miles from Luanshya. Excellent passenger trains operate on regular schedules between Cape Town and Northern Rhodesia.

Luanshya is 13° south of the equator at an elevation of 4000 ft. above sea level, and has a mild healthful climate. During the rainy season, from October to March, the rainfall averages 53 in. The area is profusely covered with vegetation. An ample supply of water for industrial and domestic purposes is available throughout the year.

The more important supplies for smelting purposes, such as wood, coal, limestone, silica, quartzite, and clay for "dolly" mud, are conveniently available. Excellent grades of coal, coke and firebrick are obtained from the Wankie Colliery Co., Ltd., Southern Rhodesia. The Wankie colliery has just completed the erection of equipment for the manufacture of silica brick. An ample supply of quartzite of good grade is available for this purpose. Silica brick is also obtained from the Belgian Congo and from India. Limestone is quarried from a large deposit a few miles from Bwana Mkubwa, which is 30 miles by rail from the smelter. An abundant supply of clay for "dolly" mud and for lining launders is available within a short distance of the smelter.

Metallurgy

Copper is contained in the ore as chalcocite, chalcopyrite and bornite. Chalcocite predominates in the eastern end of the orebody almost to the exclusion of other copper minerals. The amount of chalcocite decreases with a corresponding increase of chalcopyrite and bornite toward the western extension, where the latter predominate.

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* Superintendent of Smelter, Roan Antelope Copper Mines, Ltd.

Mining operations are, and will be for several years, restricted to the eastern extremity of the orebody. The concentrate that is produced from this ore is primarily a mixture of chalcocite (Cu_2S) and gangue material (shale). The small amount of oxides present contributes to the degree of sulfur elimination, which takes place when the concentrate is smelted, so that over 90 per cent of the iron present is converted to an oxide and enters the slag. This condition has the twofold advantage of utilizing the iron for flux and of producing a high-grade matte. The matte contains 79.00 per cent copper and only 0.5 per cent iron—almost pure Cu_2S.

Fig. 1.—General view of plant, Roan Antelope Copper Mines.

Even though practically all the iron present is available for flux, there is still a deficiency of bases for the production of a satisfactory slag. The deficiency is supplied by the addition of limestone to the charge. Limestone flux has the beneficial effect of decreasing the specific gravity of the slag and of counteracting to some degree the thickening influence of the alumina present.

As the concentrate is primarily a mixture of chalcocite (Cu_2S) and shale, a variation in the copper content has no effect on the grade of matte produced. A low-grade concentrate will produce the same grade of matte as a high-grade concentrate. The volume of slag, however, varies inversely with the concentrate grade.

As the mine is extended westward and the proportion of iron sulfide increases in the ore, the smelting operations will benefit from the increased fluxing value of the iron contained in the concentrate.

It was apparent, from the analysis of the concentrate produced by preliminary concentrating tests, that a high-grade matte and a slag abnormally high in alumina would be made in the reverberatory furnace. Many metallurgists held the opinion that considerable difficulty would be experienced in tapping, transporting and converting the matte, also that a "cold-bottom" would exist in the reverberatory furnace. The high alumina content suggested the possibility of having to provide a special method for re-treating the viscous reverberatory slag that would be produced.

Such points as these had to be determined before the design of the smelter could be completed.

Except for the practice of the Union Minière du Haut Katanga, at its Panda plant, Belgian Congo, where metallic copper is regularly tapped from the reverberatory furnaces, and at its refinery at Oolen, Belgium, where black copper is "blown" in a coal-fired converter, no accurate information concerning tapping and converting high-grade matte was available.

Preliminary smelting tests were made in a 4 by 6-ft. oil-fired reverberatory furnace. The concentrate for these tests was produced by a small pilot concentrating plant. These tests proved that:

1. Limestone was the best flux to use.
2. A calcic-silicate slag containing up to 15 per cent alumina and having a silicate degree approaching a bisilicate was sufficiently fluid to be handled and to permit satisfactory separation of the matte.
3. Matte containing 80 per cent copper could be tapped from a reverberatory furnace in the usual manner.
4. The amount of launder and ladle skulls produced would not be abnormal.
5. Matte containing 80 per cent copper could be blown in a converter without the addition of extraneous heat during the blow, but the converter would have to be heated between blows.
6. A satisfactory fuel ratio could be obtained.
7. A satisfactory recovery of copper could be obtained.

Whether or not a "cold-bottom" would form in the reverberatory furnace could be determined only from actual practice.

Flow Sheet

The general arrangement of operating and service departments provides for short, convenient means of transporting and handling products and material and allows ample space for further extension.

The coarse crushing plant and concentrator are situated near the main shaft of the mine. The ore is automatically discharged from the skips into steel receiving bins from which it is fed directly into a No. 9 gyratory crusher. The crushed ore is carried on an inclined belt conveyor

to 5½-ft. Symons cone crushers and then delivered to storage bins in the concentrator.

The concentrate, diluted to a fluid pulp (40 per cent water) is pumped through a 3-in. pipe line to the Oliver filter plant at the smelter, a dis-

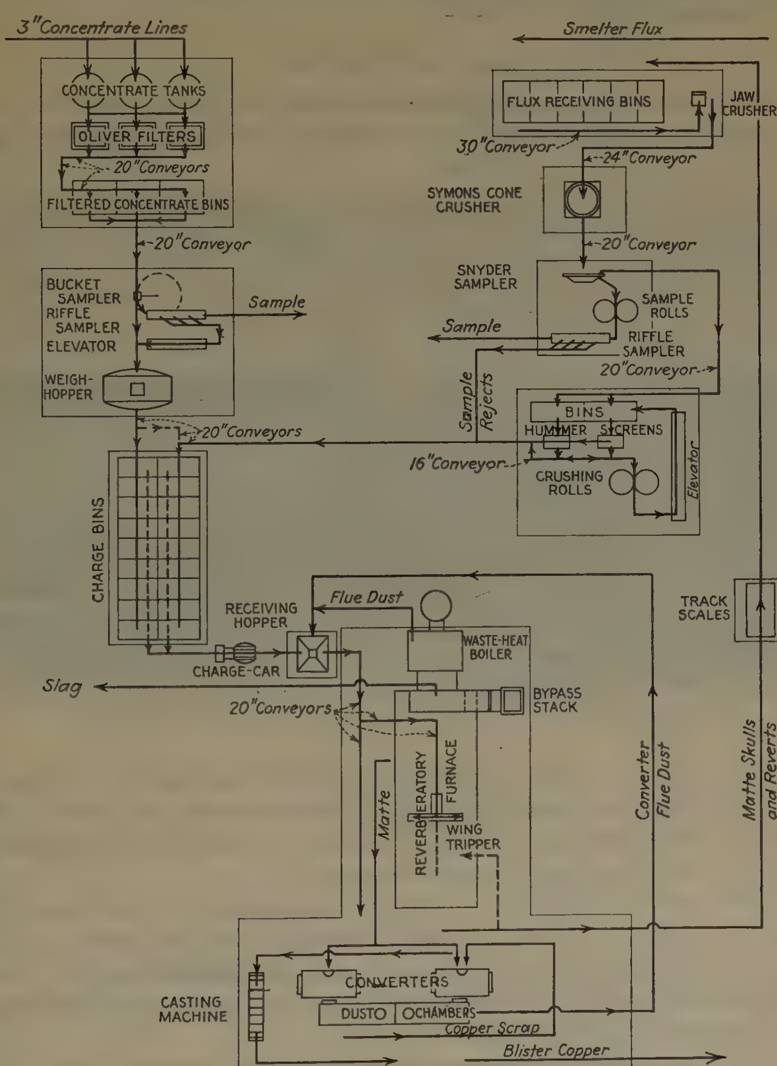

FIG. 2.—FLOW SHEET, APRIL, 1933, ROAN ANTELOPE SMELTER.

tance of 1700 ft. The filtered concentrate is delivered to a weighing machine and thence to the smelter storage bins by a system of inclined belt conveyors.

The smelter crushing plant is situated near the smelter storage bins. Crushed flux and revert material are delivered to the smelter storage bins by belt conveyors.

The reverberatory charge is transported from the storage bins to the reverberatory furnace (700 ft.) by a trolleycar specially designed to weigh and mix the ingredients of the charge.

The course of the material through the smelting plant is shown in the accompanying flow sheet (Fig. 2).

Smelting Practice

The smelter commenced operations on Oct. 15, 1931, when the first charge was fed to the reverberatory furnace. The first copper was cast

Fig. 3.—Smelting plant, Roan Antelope Copper Mines.

on Oct. 20, 1931. Except for the usual "starting-up pains" no difficulties of any moment were experienced. Early troubles were due chiefly to the inability of the operators to shut off the tap-hole and, as a consequence, several matte "runaways" were experienced.

Actual practice has demonstrated the metallurgy to be even more simple than was indicated by the preliminary smelting tests and the efficiency factors and costs to be better than the original estimates. Some of the outstanding features that contribute to the simplicity of operations, high efficiency factors and low costs are:

High Copper Fall.—The copper produced is 57 per cent of the weight of concentrate smelted or 53.8 per cent of the weight of the reverberatory furnace charge.

Fuel Ratio.—Although the charge to the furnace contains 8 to 10 per cent moisture, the fuel ratio averages 15.5 per cent. This low ratio is explained by the high proportion of matte-forming elements present in the charge.

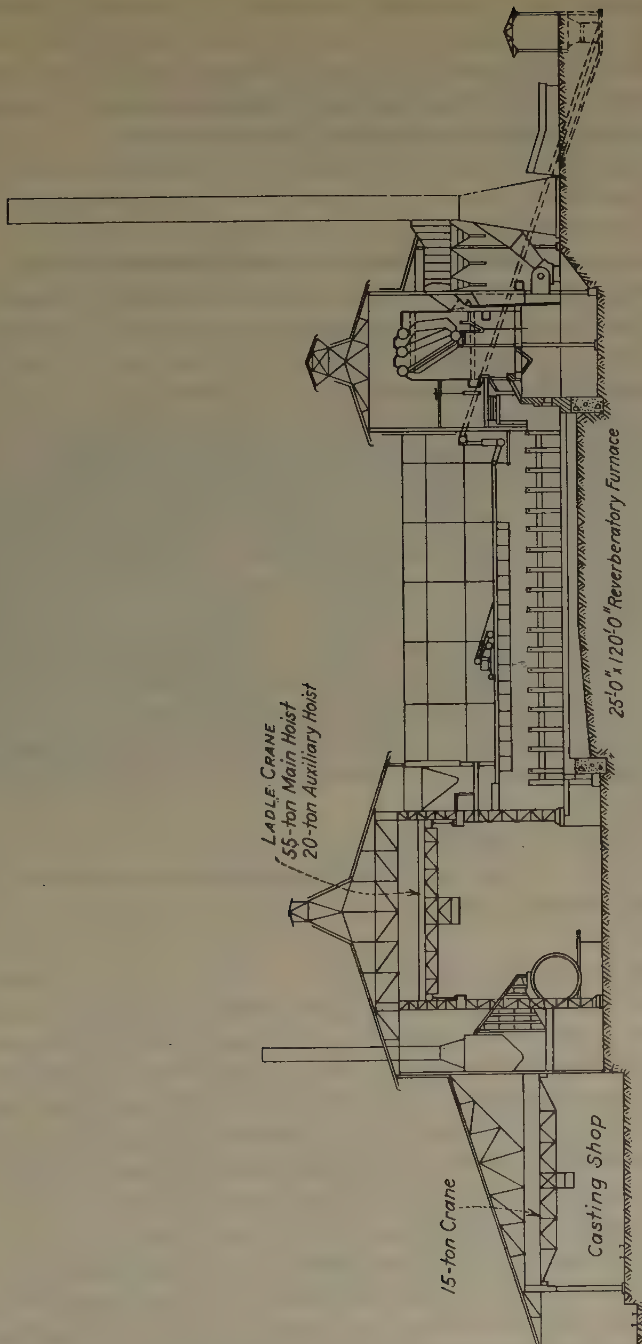

FIG. 4.—GENERAL ARRANGEMENT OF ROAN ANTELOPE SMELTER.

High Matte Fall.—The weight of matte is 70 per cent of the weight of the charge.

Low Slag Fall.—The weight of slag is 26.0 per cent of the charge.

Low Flux Burden.—The limestone required to flux the concentrate amounts to 6 per cent of the weight of the concentrate.

High Furnace Capacity.—The capacity of the reverberatory in terms of copper produced is high.

Simple Converter Operations.—The converter operation is simply the white metal or finishing stage of the usual converter practice. Hence no silica flux is required and there is no converter slag to rehandle. The consumption of compressed air is abnormally low.

High Converter Capacity.—The duration of a converter blow is short and the tons of copper produced per unit of time is high.

These factors all contribute to large plant capacity with small amount of equipment and labor.

Sampling and Weighing

The concentrate is sampled and weighed at the time it is transferred from the Oliver filter plant to the smelter storage bins. Sample cuts are taken from the stream of concentrate as it is discharged from the end of an inclined belt conveyor into the hopper of the weighing machine, by an automatic bucket sampler which is driven from the head pulley of the conveyor. The bucket sampler revolves in a horizontal plane and cuts the stream of concentrate every minute. This sample portion is automatically discharged into a vertical riffle sampler, where it is further reduced.

The discard portion of the original sample-cut is elevated by a small bucket elevator and discharged into the mouth of the weigh-hopper.

The final sample represents 0.02 per cent of the weight of the concentrate delivered.

The hopper weighing machine has a capacity to weigh 23 tons of concentrate at a time. The hopper of the weighing machine consists of a circular steel drum with dished ends and is provided with an opening in the shell through which the material is charged and discharged. The hopper is rotated by means of a worm and spur reduction gear coupled to a squirrel-cage motor of 8 hp., which is started and stopped automatically in synchronization with the starting and stopping of the feeding conveyors and ticket-printing mechanism. The hopper complete with motor and gearing is supported on a frame, which rests on weighing levers.

The entire operation of filling, weighing and discharging is controlled by an operator stationed at the scalebeam. The machine is foolproof to the extent that the operation of the feeding conveyors, weighing hopper, and ticket-printing mechanism are electrically interlocked and the weight cannot be printed until the scalebeam is in balance.

After the weight is obtained the hopper is revolved 180° and the contents are discharged into a receiving bin of 30 tons capacity. This bin is equipped with a belt feeder which delivers the concentrate on to a 20-in. belt conveyor. The capacity of the feeder and belt conveyor is sufficient to empty the bin before the next hopper-load is discharged on to it.

Twenty minutes is required to complete the cycle of loading, weighing and discharging.

Charge Preparation and Charge-car

The smelter storage bins consist of 18 steel compartments arranged in two parallel rows of 9 compartments each. The ingredients of the reverberatory furnace charge are delivered over the smelter storage bins

Fig. 5.—Oliver filter plant, smelter storage bins and crushing plant. This picture was taken from the smelter smoke stack, and shows the African bush in the background, with a bush fire at the left. During the year practically the entire bush is burned free of grass.

by two 20-in. belt conveyors and are distributed to the bin compartments by two traveling trippers, one over each row of bins.

One to three compartments are reserved for limestone flux, one is reserved for crushed reverts and the remaining bins are reserved for concentrate. Each compartment has a capacity of 2933 cu. ft. and will hold: 175 tons of wet concentrate, 160 tons of crushed limestone, or 225 tons of crushed matte. The content of each compartment is taken as a unit-lot. The weight and analysis of each lot of material is known before any of it is removed for smelting.

Each bin compartment is equipped with a motor-driven apron feeder for removing the contents. The switches are arranged within convenient reach of the operator, so that he can control all the operations of loading, weighing, mixing, traveling and discharging from his position in the cab of the charge-car.

The charge-car operates on a level track, which extends under the storage bins and over a receiving hopper at the reverberatory furnace, a

distance of 700 ft. The charge-car performs all the functions of weighing, mixing, and transporting the charge. It consists of a circular drum, 7 ft. 6 in. internal diameter and 6 ft. long, together with motor and reduction gears for revolving the drum, all mounted on a frame that is suspended from a specially designed weighing mechanism.

The drum is provided with an opening, 2 ft. long by 4 ft. 7 in., wide, upon its periphery for the purpose of charging and discharging the material. This opening is fitted with a sliding gate, operated by two independent sets of levers, one set for closing and the other for opening the gate. These levers engage projecting lugs on the gate so that it is held stationary as the drum revolves, thus enabling it to be opened for charging or discharging and to be closed when mixing.

A dial indicates the weight of material in pounds, as the material is fed into the drum. A typical charge consists of: 18,800 lb. wet concentrate, 1128 lb. limestone flux, and, when crushed matte is available,

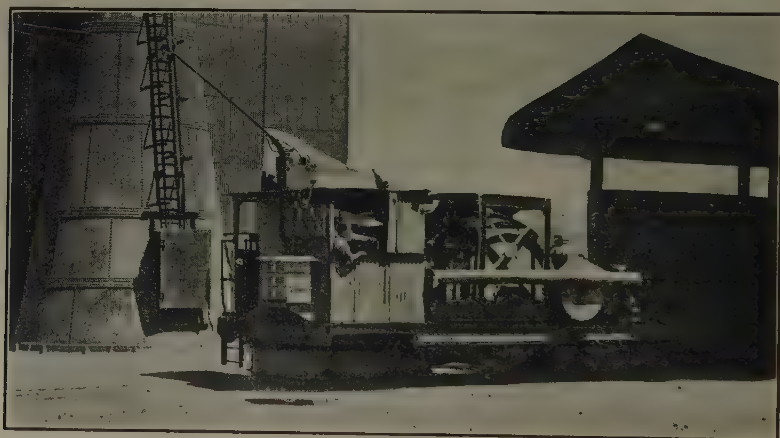

FIG. 6.—CHARGE-CAR.

17,300 lb. wet concentrate, 1050 lb. limestone flux, and 1650 lb. crushed matte skulls. This fills the drum a little more than half full, which allows sufficient empty space for proper mixing.

A system of motor-operated cams is provided for lifting the frame and its load, so that there is no weight on the knife-edges when weighing is not being done.

The drum is revolved and its contents mixed during the time the car travels from the storage bins to the reverberatory furnace.

SMELTER CRUSHING PLANT

The equipment of the smelter crushing plant consists of six storage bins, one 24 by 15-in. jaw crusher, one 5½-ft. Symons cone crusher, and one set of 42 by 16-in. rolls in closed circuit with two 4 by 5-ft. Hum-mer

vibrating screens, all arranged in the order mentioned and provided with the necessary feeders and conveyors. The crushed product is conveyed to the smelter storage bins by means of a 20-in. inclined belt conveyor.

Limestone flux is crushed to pass a 4-mesh screen. Smelter reverts are crushed to ⅞-in. size—the product of the Symons crusher.

The plant has capacity to crush, in 8 hr., a supply of limestone flux sufficient for 10 days' smelting operations.

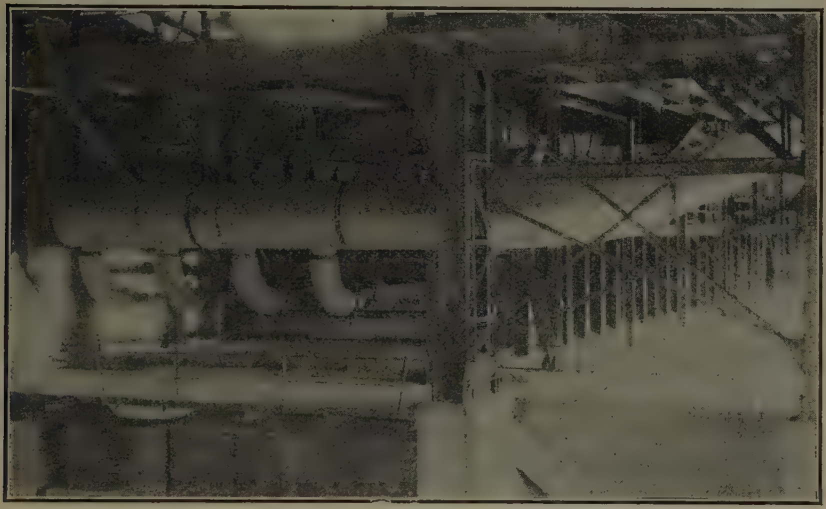

FIG. 7.—BURNER END OF REVERBERATORY FURNACE.

A sampling plant is situated between the Symons crusher and roll crushing plant and consists of a 60-in. Snyder sampler, a set of 24 by 14-in. rolls, and a vertical riffle sampler.

Reverberatory Smelting

The reverberatory furnace is built with silica brick and is 25 ft. wide and 100 ft. long, inside dimensions. (The furnace as originally constructed was 120 ft. long.) The side walls are 24 in. thick and 7 ft. 6 in. high. The arch is 18 in. thick for a distance of 65 ft. from the burner wall and 15 in. thick for 26 ft. over the settling zone. The radius of the arch is 35 ft. The 15-in. arch was originally constructed with a droop of 3 ft. in 38 ft. It was thought that a droop in the arch would be necessary to maintain sufficient temperature over the surface of the bath to keep the high-alumina slag in a fluid condition. Later it was found that the droop was unnecessary, so at the time of the first arch repair, when the uptake area was increased from 64 sq. ft. to 96 sq. ft., the droop was removed. The 18-in. arch is provided with ribs (12 in.

high and 15 in. wide) spaced at 5-ft. intervals. When the brickwork of the arch has been corroded to a thickness of about 10 in., the panels between the ribs are filled in with silica brick, thus providing an additional 12-in. thickness to the roof, and consequently a longer furnace campaign.

The usual method of tying the furnace is used: 15-in. box-type buckstays held at the bottom by concrete abutments and tied across the top of the furnace with 2-in. dia. tierods are spaced at 5-ft. intervals along the side walls. The furnace is tied longitudinally by four 3¾-in. dia. tierods, two at each side. The end buckstays are similar to those along the side walls. Cast-steel skewback plates are bolted to the buckstays to form a continuous tie between the buckstays, the full length of the furnace.

The furnace is heated with pulverized coal. The combustion mixture is blown into the furnace through four 12-in. dia. burner pipes, spaced at 4-ft. centers in the end wall. The outside burner pipes are 6 ft. 6 in. from the side walls.

A central pulverizing plant situated 800 ft. east of the smelter site furnishes the powdered coal for use in the direct-fired boilers at the power plant and for heating the reverberatory furnace, converters and casting drum at the smelter. The pulverizing equipment consists of: one "Pennsylvania Hammer" coal crusher, two coal-fired rotary driers, and three 8 by 5-ft. "air-swept" Hardinge ball mills coupled with "superfine" classifiers and a cyclone dust collector. The powdered coal is transported through a 6-in. pipe line to the point of consumption by two 6-in. Fuller-Kinyon pumps.

The coal is pulverized to a fineness of 80 per cent through a 200-mesh screen. The coal and ash analyses are as follows:

Coal Analysis	
British thermal units	13,640
	Per Cent
Hydrogen	3.8
Volatile carbonaceous matter	27.4
Fixed carbon	62.1
Sulfur	1.16
Ash	9.5

Analysis of Ash	Per Cent
SiO_2	48.62
FeO	6.60
CaO	6.12
Al_2O_3	33.96
MgO	0.47

The powdered coal is delivered from the 6-in. pipe line into four steel bunkers situated over the firing end of the reverberatory furnace. Each bunker has a storage capacity of 18 tons of powdered coal. The operator at the pulverizing plant controls the distribution of the coal by means of electrically operated valves placed in the pipe line. A paddle, suspended in each bunker, operates a signal light at the pulverizing plant to warn the operator when a bunker is full.

Each coal bunker is equipped with a Bailey feeder which is operated by a 2-hp. motor through variable-speed reduction gears and tex-rope drive.

The coal drops from the feeders into the stream of primary air through 3-in. pipes connected to 7-in. injectors, which are situated in the primary air lines, and thence to the burners, where it joins the supply of secondary air.

The primary air is supplied by four fans, each of which has a capacity to produce 1100 cu. ft. of free air per minute at 9-in. water-gage pressure. The secondary air is supplied by a centrifugal forced-draft fan of 27,000 cu. ft. per minute capacity. The secondary air is blown through an Usco air preheater situated in the flue system between the boiler and the stack. The air preheater consists of 70 elements, which provide a total heating surface of 10,500 square feet.

The hot secondary air is conducted through a 4-ft. dia. insulated conduit to the burner end of the furnace, where it is distributed by means of butterfly valves placed in the branch lines leading to the burners. The temperature of the secondary air as it leaves the preheater is 450° F. and is 400° F. as it is delivered to the burners.

A balanced combustion is maintained in the furnace at all times. The draft is kept at 0.025 in. of water. Gas analyses are obtained from samples aspirated through a hole in the side wall, at a point near the uptake or exit of the furnace. The combustion mixture is adjusted to produce a furnace gas of the following analysis: CO_2, 18.2; O_2, 0.6; CO, 0.0 per cent.

Charging the Furnace

The mixed charge is transported from the receiving hopper to the charge floor of the furnace by a system of belt conveyors, which are electrically interlocked and operated by a push-button control from the charge floor. The charge is distributed by a two-wing tripper operating in conjunction with a 20-in. conveyor which extends the full length of the furnace.

A small hopper, suspended under the head pulley of the charge conveyor and equipped with three discharge pipes, provides means for fettling the burner wall.

The charge is fed into the furnace through 6-in. pipes which extend vertically downward from the overhead hoppers into holes in the roof adjacent to the side walls. There are 12 charge pipes on each side spaced at 5-ft. intervals. Leakage of air into the furnace around the charge pipes is prevented by a copper casting snugly fitted to the charge-pipe and well grouted to the surface of the furnace arch by a thick pad of fireclay mortar. Cast-iron charge-hole castings were used at first, but these castings soon burned out and were abandoned in favor of the present arrangement.

The wet charge is fed into the furnace through 6-in. pipes without any trouble from choking. It is the practice to feed the charge into the

furnace directly as it is discharged from the tripper and not allow it to accumulate or remain in the hoppers to dry. In this way the amount of dusting in the furnace is minimized and the formation of cuprous oxide in the charge diminished. The extreme fineness of the concentrate, together with the high copper content, promotes oxidation. Concentrate that has been kept in storage or has been allowed to dry shows an appreciable increase in the percentage of cuprous oxide present.

The atmosphere around the furnace and on the charge floor is free from dust and gas. The working conditions are excellent.

Even though the charge is wet there is an appreciable amount of dusting in the furnace. The dust that is recovered from the boiler hoppers and flue system amounts to 3 per cent of the weight of the charge and contains 31 per cent copper in the form of cuprous oxide and copper sulfate. The cuprous oxide in the dust that impinges against the end and uptake walls of the furnace causes rapid corrosion of the silica brick of these walls. During the first campaign of the furnace the end wall had to be replaced three times within a period of four months. Water-cooled copper blocks (18 by 18 by 3 in.) were incorporated in the brickwork of the end wall. This wall lasted seven months. It is now proposed to face the uptake walls with magnesite brick. A panel of magnesite brick 2 ft. wide was built into the face of the end wall and has suffered no corrosion after three months' operation.

TABLE 1.—*Reverberatory Charge*

	Analysis, Per Cent							
	Units	Cu	SiO_2	FeO	CaO	Al_2O_3	MgO	S
Concentrate	100	58.67	13.4	3.90	0.10	4.12	0.70	16.02
Limestone	6		0.6	0.17	52.30	1.02	2.04	
Matte reverts	3.6	78.73		0.58 (Fe)				19.52
Reverberatory flue dust		31.22	19.82	4.59	3.13	11.37	0.35	5.7
Converter flue dust		73.29	0.88	0.69	0.22	1.32		11.41

An analysis of the reverberatory charge is given in Table 1. Flue dust is not mixed with the charge but is fed to the reverberatory at irregular intervals.

Reverberatory Slag

Experiments have proved that the most economical balance between the concentrator and smelter operations is obtained with a concentrate approximating 58 per cent copper. The concentrator department can make a better recovery of copper by producing lower grade concentrate, but this is offset in the smelter operations, by: (1) high copper loss to the slag due to the increased amount of slag; (2) greater amount of flux

required; (3) greater amount of barren material to smelt, involving the consumption of more coal and a decrease in smelting capacity.

The reverberatory slag is skimmed through an opening in the end wall of the furnace. A second skimming opening has been provided in the side wall for greater convenience in making protective slag linings in the ladles that are to be used for handling molten copper. As there is no converter slag available, reverberatory slag is used for this purpose.

The sills of the two skimming doors are 24 in. above the level of the lower tap-hole. The surface of the slag is raised 6 to 12 in. above the level of the sills by temporary clay-mud dams built across the openings. The total depth of bath varies between 24 and 36 in., of which 12 to 24 in. is matte. The smelting rate and fuel efficiency are better when the depth of molten matte is maintained at 12 or more inches.

The molten slag is laundered into cast-steel ladles of 200 cu. ft. capacity. The ladles are carried on specially designed trucks, which are equipped with electrically operated means for tipping the ladle. The electrical connection is made by means of a jumper-cable from the trolley circuit.

The slag is hauled to the dump by a gasoline-electric locomotive. This locomotive has been modified to operate from an overhead trolley circuit when hauling slag and by its own gasoline power plant when operating over tracks not equipped with the trolley line.

The slag analysis is: SiO_2, 47.3 per cent; FeO, 13.7; CaO, 11.4; Al_2O_3, 14.7; MgO, 2.9. This will be recognized as an unusual analysis for slag produced in a reverberatory furnace. The iron content is low and the alumina content is abnormally high.

The alumina is figured as a base and the charge is calculated to produce a slag having a silicate degree between a sesquisilicate and a bisilicate. This high-alumina slag has a relatively low formation temperature and is sufficiently fluid at the existing temperature (2100° F.) to permit separation of the matte and to flow readily from the furnace.

It is customary to keep the silicate degree as high as practical in order to reduce the consumption of limestone flux and fuel to a minimum and to produce the minimum amount of slag. Practice has demonstrated that the most economical slag is one whose silicate degree approximates 1.75. Above this point of acidity the slag is too viscous. As the acidity is decreased the slag becomes more fluid and increasingly corrosive to the silica brick.

The actual amount of copper lost in the viscous slag of high copper analysis and low volume is less than is lost in the fluid slag of lower copper assay and larger volume. The cuprous oxide present in the concentrate (Cu_2O, 2.40 per cent) contributes to the loss of copper in the slag. By adding fine coal to the charge (3.0 per cent of the weight of concentrate) the percentage of copper in the slag has been reduced from 2.25 to 1.20 per cent Cu.

Matte

Analysis of the matte is: Cu, 78.73 per cent; Fe, 0.58; S, 19.52; insoluble, 0.14.

The matte is tapped through a 3-in. tap-hole situated in the side of the furnace, 75 ft. from the burner wall. The tapping block consists of a magnesite brick cast in a block of copper. Two 3-in. holes at 9-in. centers are drilled through the copper and the brick, after the block has been cast. This provides for an upper and a lower tap-hole in the same block.

In order to prevent the formation of heavy matte skulls, the ladles are filled as quickly as possible. When the matte flows too slowly from one

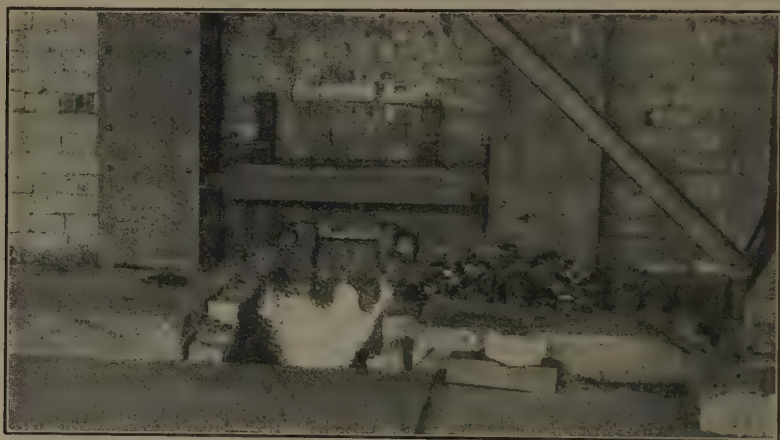

FIG. 8.—TAPPING MATTE.

tap-hole, both tap-holes are opened. A ladle of 22 tons capacity is ordinarily filled in less than 10 minutes.

The high-grade matte is tapped without difficulty; in fact, when taps are made at frequent intervals the matte will follow a tapping bar as it is withdrawn from the tap-hole.

The tap-hole is closed with a clay dolly in the usual manner, and is opened by burning out the hole with oxygen. A tapping bar is not usually employed. The matte is laundered into cast-steel ladles supported on low four-wheel trucks which operate on a track parallel to the side of the furnace and extending into the converter aisle. The ladle trucks are moved by means of a cable operated from an electric hoist situated on the far side of the converter aisle. Every facility has been provided to expedite transfer of the molten matte from the reverberatory furnace to the converters. The average weight of the skulls produced is 5 per cent of the weight of the matte. The same ladle is used several times before it is necessary to dump the frozen shell.

A Stirling boiler, closely connected to the uptake of the reverberatory furnace, and situated directly in front of the furnace, utilizes 52 per cent of the heat value of the coal burned. The temperature of the furnace gas is 2000° F. at the inlet and 620° F. at the outlet of the boiler.

The hot gas, after leaving the boiler, is drawn through an Usco air preheater by a centrifugal induced-draft fan (67,000 cu. ft. per minute capacity) and is discharged through a short connecting flue into the reverberatory stack. A direct bypass flue connection with the stack is provided for use when the air preheater and induced-draft fan undergo cleaning or repairs.

When the waste-heat boiler has to be shut down for cleaning or for repairs the furnace gas can be directed into a bypass stack which is situated at the end and side of the furnace. This is a firebrick stack, 6 ft. square by 75 ft. high. Water-cooled dampers situated in the flues provide means for switching the furnace gas to either the boiler or bypass stack.

The main stack consists of a steel shell 9 ft. in diameter and 180 ft. high, lined with 4½ in. of firebrick.

Operating data as of February, 1933, are given in Table 2.

TABLE 2.—*Operating Data* (*February*, 1933)

Dry charge per furnace day, tons	340
Coal consumed per day, tons	52.7
Fuel, per cent of charge	15.5
Flux, per cent of concentrate	6.5
Flux, per cent of charge	5.09
Slag, per cent of charge (excluding reverts)	25.5
Slag, per cent of charge (including reverts)	21.22
Silicate degree of slag	1.704

CONVERTING

There are two 12 by 20-ft. basic-lined Peirce-Smith type converters. Each is equipped with thirty 1½-in. tuyeres spaced at 6-in. intervals. One converter has sufficient capacity to treat all the matte produced by the reverberatory furnace, the other converter is used as a spare.

The converters are revolved by means of 35-hp. Holman air motors operating through reduction gears. Compressed air for operating the motors is supplied at 80 lb. per square inch pressure from a receiver that is connected through a check valve with the industrial compressed-air system. An automatically operated air compressor functions as an auxiliary unit to maintain the air supply in the receiver at the required pressure.

The advantage of an air motor over an electric motor for revolving a converter is obvious. The operation is divorced from troubles resulting from sudden failures of the electric power. A sufficient volume of air at proper pressure is always available in the receiver to revolve the

converters into a safe position in case of failure of the supply of air to the tuyeres.

Compressed air for converting purposes is supplied by a motor-driven four-crank single-stage vertical compressor, which has a rated capacity to deliver 15,000 cu. ft. of free air per minute at 15 lb. per square inch pressure. The compressor is in the power house and delivers its air to the smelter through a pipe line which is 30 in. in diameter and 115 ft. long.

One overhead crane serves all converter aisle operations. It has a span of 57 ft. 6 in., and consists of two bogies operating independently of each other. The large bogie carries the motor and winding drum for the main hooks and operates on a track attached to the top surfaces of the crane girders. The steel ropes extend outside the crane girders to the bale, which carries two hooks properly spaced to connect with the lugs on each side of the ladles. This unit has a lifting capacity of 50 tons. The small bogie carries the motor and winding drum for the auxiliary hook. It operates on rails attached to the lower inside flanges of the crane girders. This arrangement makes it possible to use the auxiliary hook on either side of the main hooks. This unit has a lifting capacity of 20 tons.

Each converter is provided with a separate dust chamber and stack. An average of 40 tons of flue dust per month is recovered from the dust chambers. Analysis of the flue dust is: Cu, 73.29 per cent; SiO_2, 0.88; FeO, 0.69; CaO, 0.22; Al_2O_3, 1.32; S, 11.41.

As previously stated, the matte contains 78.7 per cent Cu, 0.58 per cent Fe, 19.52 per cent S, 0.14 per cent insoluble—almost pure copper sulfide. The operation of converting such a matte is identical with the white-metal or finishing stage of the usual practice of converting lower grade matte.

Thermal calculations show the heat generated by the oxidation of the sulfur to be slightly in excess of the heat dissipated during the operation. This is corroborated by temperature readings taken at intervals during the converter blow. The temperature of the empty converter just before charging three ladles (60 tons) of matte was 2230° F. The temperature of the matte was 2010° F. The temperature increased during the blow at the gradual rate of 2.3° F. per minute. At the finish of the blow the temperature reading was 2370° F. These temperatures agree closely with those obtained in the finishing or white-metal stage when converting lower grade matte.

When the converter was designed strict attention was paid to the question of conservation of heat. The mouth was made as small as was considered practical (4 ft. 6 in. by 5 ft. 4 in.) and the magnesite brick lining was backed with an insulating layer of fireclay brick.

The original linings consisted of 9 in. of magnesite brick backed with a 4½-in. layer of fireclay brick and separated from the shell by a 2-in. pad

of crushed dead-burnt magnesite. Along the tuyere belt and for 10 courses (30 in.) above it the magnesite lining was 16 in. thick. It is planned to replace the 9-in. magnesite brick with 12-in. brick as the linings require repair.

The converters are heated by means of powdered coal introduced through a 14-in. burner port situated in the end wall. The products of combustion exit through the mouth to the converter flue system. Heating is done only between blows and the burner port is sealed with clay-mud during the blow.

When the blow is finished and the converter emptied, the burner port is opened by removing the mud seal. The fuel is then turned on and the converter charged with cold copper scrap. From 5 to 15 tons of cold copper scrap can be melted between blows. After the copper scrap has melted, the converter is recharged with matte.

The usual converter charge consists of current copper scrap and three ladles (60 tons) of molten matte. It is the practice when four ladles (80 tons) of molten matte is blown to charge the fourth ladle when the blow is half finished, after blowing for about one hour.

As previously mentioned, the ladles in which the molten copper is handled are lined with a shell of reverberatory slag. A considerable portion of this lining adheres to the copper skull which is remelted in the converter. This slag acts to flux the small amount of iron present in the matte. If there happens to be insufficient slag for this purpose, and the converter becomes dirty from accumulated iron, 4 or 5 tons of molten reverberatory slag is added and blown with a straight charge of matte. This very effectively cleans the converter.

The small amount of converter slag that is produced is allowed to remain in the ladle to form an insulating blanket or crust over the molten copper during the casting operation.

It requires about 2 hr. and 25 min., depending on the condition of the tuyeres and the temperature of the charge, to finish 60 tons of matte to blister copper. During this period the tuyeres are punched almost continuously. The finish is determined by the appearance of the flame and the copper on the rod.

The consumption of magnesite brick for lining purposes averages 2.47 lb. per ton of copper produced. The consumption of magnesite is represented almost entirely by the repairs to the tuyere belt. The life of the lining at this point has averaged 4000 short tons of copper produced. The damage to the tuyere belt is caused chiefly by the mechanical action of reaming the tuyeres rather than by the action of corrosion.

The corrosive action of the coal ash on the end wall opposite the burner has necessitated renewal of the brickwork after 17 months' operation. As no converter slag is produced, there is no iron available to make a protective coating of magnetite over the magnesite lining, as is customary

in the usual converter practice. An attempt was made to provide the protective coating of magnetite by adding a small amount of crushed hematite to the converter charge. The scheme was successful but had to be discontinued on account of the trouble caused in the reverberatory furnace by the excess magnetite, which accumulated in the form of a blanket between the slag and the matte. The calcic-silicate slag would

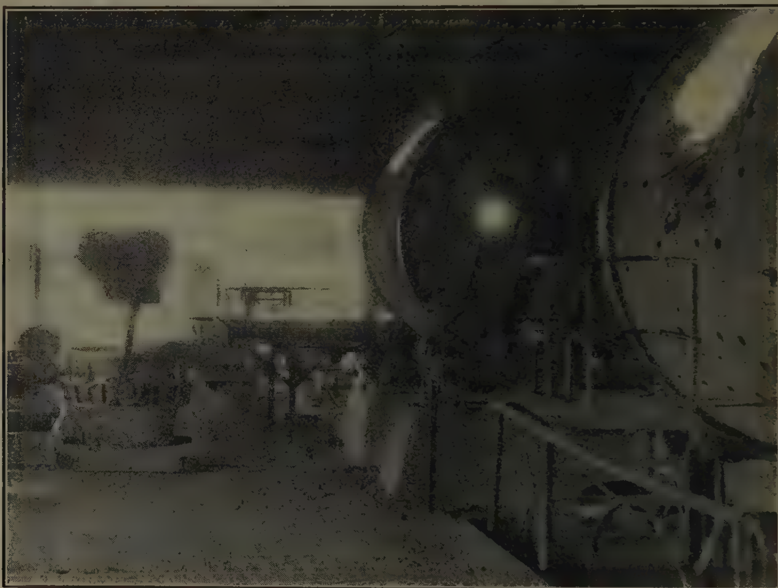

FIG. 9.—CONVERTERS.

carry no magnetite with it; the high-grade matte had no affinity either chemically or mechanically for it. As a result, every pound of magnetite that was charged into the reverberatory furnace with the converter reverts remained there and had to be removed with rabbles.

Converter data are listed in Table 3.

TABLE 3.—*Converter Data*

Charge, tons matte	60
Air pressure, lb. per sq. in	15
Air consumed per ton of matte treated, cu. ft	30,590
Free air consumed per minute, cu. ft	13,700
Minutes per converter blow	134
Minutes per ton of matte treated	2.23
Minutes per ton of copper produced	2.83
Blister produced per blow, tons	47.5
Blister produced per converter hour, tons	21.3
Fuel consumed per day, tons	5.9
Fuel consumed for heating converter and melting scrap, lb. per hour firing	690
Analysis of blister copper, per cent Cu	99.6

Casting

The blister copper is cast into 350-lb. cakes (18 by 28 by 3 in.) on a straight-line casting machine. The machine is equipped with 50 molds and extends from the converter aisle into the casting aisle. A 20-hp. motor drives the machine through a series of reduction gears.

The copper is poured from a ladle, suspended from the main hooks of the converter aisle crane, into a casting drum. The casting drum consists of a brick-lined cylinder 4 ft. in diameter and 10 ft. long, supported at

Fig. 10.—Casting copper.

right angles over the line of molds on rollers. A hole, 24 in. wide and 36 in. long, is provided in the shell to receive the stream of molten copper, and to allow for exit of the products of combustion. Heat is supplied by means of powdered coal blown through a burner port situated in the head of the cylinder.

The copper is discharged from the cylinder into the molds through a 5 by 9-in. horizontal slot to which a short lip is attached (Fig. 10). When the lip is in its lowest position it clears the top of the molds by one inch.

The cylinder is revolved by means of an arm fixed to the end plate and a cable from the end of the arm to a motor-driven winding drum. A counterweight suspended from the end of the arm provides the means for revolving the cylinder in the opposite direction. Before the adoption of this method of casting, an open spoon was used, which caused an abnormal amount of copper scrap to be produced. The present arrangement has greatly reduced the amount of scrap.

Reject cakes and fins are charged into the casting drum to control the temperature of the copper as it is cast.

The cast copper cakes are cooled by water sprays as the molds move toward the discharge end of the machine. The cakes are discharged from the molds on to an inclined ramp (Fig. 11) from where they are trucked by hand, finned and loaded into standard railway gondolas for shipment.

FIG. 11.—DISCHARGE END OF CASTING MACHINE.

The molds are made by casting low blister or clear copper in demountable master molds directly from a ladle suspended from and tilted by the converter crane.

One side of the mold is extended to form an overlapping lip which covers the space between molds on the casting machine and prevents spills.

A cast-iron pouring plate containing the Roan Antelope brand mark "R.A." is cast or embedded in the bottom of each mold. When a copper mold is worn out, these plates are recovered and used again.

TABLE 4.—*Casting Data*

Capacity of casting machine, tons per hour	25
Average weight of cakes, lb	350
Coal consumed per day for heating casting drum, tons	1.25
Fins, per cent of copper cast	1.20
Molds consumed, per cent of copper cast	1.00
Average analysis of blister copper cast, per cent Cu	99.6

The copper is sampled by the splash or shot method from the stream as it flows from the lip of the ladle. The sample is screened to remove the oversize pieces.

Casting data are given in Table 4.

Recovery and Plant Capacity

The average over-all plant recovery is 97.92 per cent. The losses are distributed as follows:

	Per Cent
Reverberatory slag	0.60
Reverberatory stack	0.85
Converter stack and unaccountable	0.63
Total	2.08

Fig. 12.—Native furnace crew.

The smelter has never been required to operate at its full capacity. The potential capacity of each unit is estimated to be:

	Short Tons Blister Copper per Month
Reverberatory furnace	5900
Converters:	
Using one converter, four blows per day	5700
Using two converters and allowing time out for repairs	8650
Casting:	
Allowing time for repairs, replacement of molds and cleaning machine	9000

Personnel

In all, 39 Europeans and 150 natives are employed in smelting operations.

Very few of the European employees had even seen a smelter previous to 16 months ago, when operations commenced. Today, they perform the work like veterans.

The efficiency of the native laborers has increased amazingly. Many of them had never before seen a white man, much less a smelter. The white man's tools were strange contraptions to them. They had to be taught to use a wheelbarrow and they soon discovered it was easier to push it than to carry it on their heads. From this condition of abject ignorance the natives are developing rapidly into reliable workmen. They can be depended upon to do well the routine tasks which require no personal judgment.

Good food, sanitary conditions and excellent medical attention have greatly improved the physical condition of the native. The "raw boy" direct from the "bush" is usually undernourished, owing to a life-long diet of "mealie meal" (ground corn). That the natives appreciate the personal comforts earned through labor is demonstrated by the small labor turnover and the large number of applications for employment.

Copper-refinery Waste-heat Boilers at Great Falls Reduction Department, Anaconda Copper Mining Company

By E. S. Bardwell,* Great Falls, Mont.

(New York Meeting, February, 1934)

Each of the three refining furnaces in use at Great Falls is provided with a waste-heat boiler. The general arrangement of furnace and boiler is as shown in Fig. 1. Two of the furnaces have hearths 45 ft. 7 in. by 14 ft. 6 in.; the third has a hearth 46 ft. 10½ in. long and of the same width as the other two. The furnaces at present are equipped for firing with natural gas. As originally constructed two of the furnaces had hearths 36 by 14 ft., and were designed to use coal on grates, but these furnaces were lengthened and fired successively with pulverized coal and oil. The third furnace was added in 1926 and was oil-fired until 1928. Since that year natural gas has been in use on all of the furnaces. The changes that have occurred in connection with furnace design and the comparative performance of the furnaces with various fuels were described[1] by the author in 1932.

Two of the furnaces are provided with Stirling class A-30 boilers, rated at 400 hp. and having 4008 sq. ft. of heating surface. The third furnace has a class A-21 Stirling waste-heat boiler rated at 300 hp. with 2805 sq. ft. of heating surface. The boilers are not baffled.

The waste-heat boilers are placed on a platform 17 ft. 3 in. above the level of the furnace charge floor. The waste gases from the furnace pass by a vertical uptake flue, 4 by 3 ft. inside at the point where it enters the chamber, to a brick chamber from which the gases pass over a bridge wall and into the boiler setting. The uptake flue and chamber are constructed of silica brick.

Each furnace has a steel stack 125 ft. high, measured from the boiler floor. The stacks have a 4½-in. firebrick lining and are 5 ft. in diameter

[A reverberatory furnace conference, attended by representatives of the various companies operating reverberatory furnaces, was held at Salt Lake City in May, 1930. During the discussion various figures were quoted as to the yield secured in the generation of steam from the waste gases from reverberatory furnaces. There was such a wide variation that the Institute Subcommittee on Pyrometallurgy of Copper Ores decided that it would be worth while to secure a symposium of papers describing the waste-heat boiler practice at as many smelters as possible. This paper and the following papers on waste-heat boilers are presented in accordance with this plan.—Ed.]

* Superintendent of Copper Refineries, Anaconda Copper Mining Co.

[1] E. S. Bardwell: A Comparison of the Use of Various Fuels in Copper-refining Furnaces. Page 449, this volume.

measured inside the lining. For emergencies each furnace is provided with a steel bypass flue lined with firebrick. The inside diameter of the bypass flue is 5 ft. The boiler setting consists of 8 in. of common brick with a 9-in. firebrick lining. The breaching leading from the boiler setting to the stack enters the stack 17 ft. above the boiler-floor level. Feed-water pumps are on the boiler platform in the space between the boiler setting and the stack. The boilers ordinarily are operated at 75 lb. pressure with feed water at 60°F. The draft in the uptake normally amounts to 0.4 in. of water.

In considering the efficiency of waste-heat boilers it should always be kept in mind that the prime object of burning fuel in a metallurgical

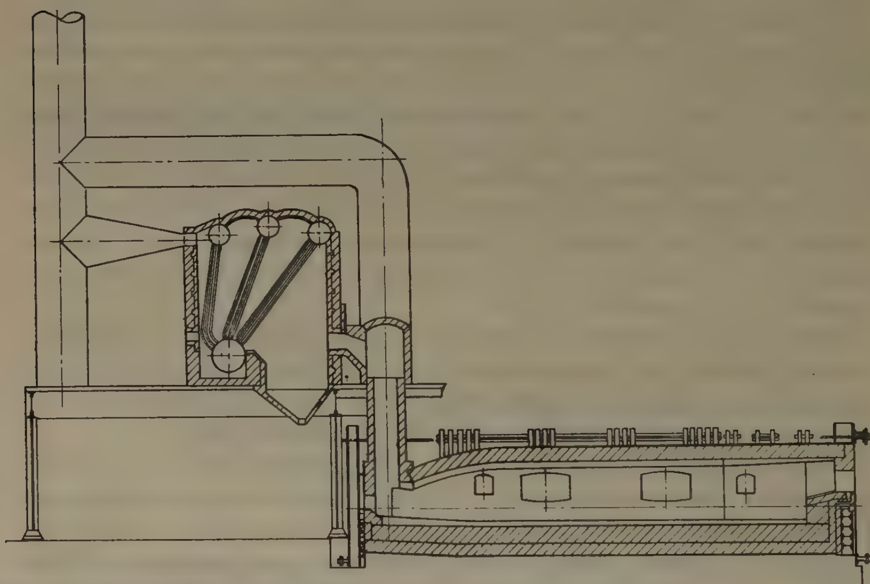

FIG. 1.—GENERAL ARRANGEMENT OF FURNACE AND WASTE-HEAT BOILER, GREAT FALLS REDUCTION DEPARTMENT.

furnace is to smelt charge and not to produce steam. It is always possible to so operate a furnace equipped with waste-heat boilers as to increase the rate of steam production, but this invariably will mean less favorable fuel ratio as far as the smelting operation is concerned. For a copper-refining furnace such a method of operation is entirely out of the question. From 65 to 75 per cent of the total fuel is consumed during about one-half of the furnace cycle, which ordinarily is 24 hr. During this period the charge is being melted and we wish to keep the furnace atmosphere slightly oxidizing. An attempt to force the furnace inevitably results in imperfect combustion in the smelting chamber, loss of time and less favorable fuel ratios, even though more steam may be

produced. The true function of a waste-heat boiler is just what the name implies; it is an apparatus designed to recover from waste furnace gases, for use in other operations, the maximum amount of heat that is economically practicable. If additional steam over and above that available from waste-heat boilers is required, it is much more economical to produce it in direct-fired boilers.

This may seem so self-evident as not to be worthy of mention. It is, however, being brought out in order to emphasize the fact that the efficiencies of waste-heat boilers should not be compared apart from the fuel efficiencies obtained in the furnaces with which the waste-heat boilers are connected. For example; two tests made on a certain waste-heat boiler, using in one case pulverized coal and in the other case oil, showed efficiencies of 38.9 and 40.2 per cent respectively based on the fuel consumed in the furnace. This might lead to the erroneous

TABLE 1.—*Rate of Steam Production*

	With Oil Fuel, 400-hp. Boiler	With Gas, 400-hp. Boiler	With Gas, 300-hp. Boiler
Charging	295	178	146
Melting	395	290	306
Skimming		310	283
Rabbling	288[a]	298	134
Poling	368	573	630
Casting	260	441	282
Average	337	322	312

[a] Includes skimming.

conclusion that with pulverized coal as fuel the boiler was not as efficient as when oil was used. In this test, however, it was found that 46.7 per cent of the heat value of the pulverized coal was absorbed in the furnace, whereas with oil 44.7 per cent only was so absorbed. The stack loss amounted to 14.4 per cent with pulverized coal as against 15.1 per cent with oil. In either case it would have been possible to have so operated the furnace that a larger proportion of the total heat value of the fuel could have been recovered in the waste-heat boilers, but such a method of operation would be uneconomical, for reasons that have been pointed out. The true efficiency of a waste-heat boiler would be represented by the quotient of the heat units absorbed divided by the heat units contained in the waste gases from the furnace. Efficiencies based on the fuel supplied to the furnace can only be properly interpreted in cases where a complete heat balance is available. In cases where considerable heat may be developed from reactions taking place in the smelting chamber a complete heat balance is even more necessary.

Three fairly complete tests have been conducted on the waste-heat boilers at the furnace refinery. One in 1924 when oil was being used as fuel and two after natural gas had been substituted. Of the latter, one was made on one of the furnaces equipped with a 400-hp. boiler; the other on the furnace equipped with a 300-hp. boiler. Each of these tests extended over a complete furnace cycle—approximately 24 hr. The average rate at which fuel was supplied was measured, as was also the average rate of evaporation, to which corrections were applied for blowoff. The average rate of steam production during the furnace cycle, expressed in boiler horsepower, is shown in Table 1, page 227.

Repeated observations have shown that the average temperature of gases entering the waste-heat boilers is 1940° F. As soon as we changed from oil to natural gas we discovered that, based on the metallurgical calorific value of the fuel used, we required more British thermal units when natural gas was used than when the fuel was oil. We also noted that the steam production did not show a corresponding increase. For example, with a charge of the same size we would burn an average of 31,117 cu. ft. of gas per hour as against 1465 lb. of oil, or, based on the metallurgical calorific value of the two fuels, approximately 10 per cent more heat units per hour were required with gaseous fuel than with oil. With oil the gases leaving the waste-heat boilers averaged about 600° F., while when natural gas was used as fuel the gases averaged about 800° F.

Calculations based on the analyses of the respective fuels show than when oil is being used we have 1,875,200 cu. ft. of gas per hour going to the waste-heat boilers while with gas the volume has increased to 2,140,258 cu. ft. per hour. When oil is used the products of combustion entering the waste-heat boilers contain an average of 15,572,950 B.t.u. per hour, while with gas this is increased to 17,954,509 B.t.u. per hour. The increased gas volume and increased heat content of these gases resulting from the use of natural gas explains the increased temperature of the gases entering the stack.

The gases entering the stack when oil is being used as fuel average 4,261,681 B.t.u. per hour and when gas is being used, 6,627,921 B.t.u. This leaves available for producing steam and caring for radiation and convection losses in the case of oil 11,311,265 B.t.u. and with gas 11,326,588 B.t.u.

From the last-mentioned figures we may subtract 760,000 B.t.u. per hour, which has been computed to represent radiation and convection losses, and dividing by 33,305, arrive at 317 boiler-hp., which is very close to the figures arrived at through our tests.

The figures given above, as has been stated, are based upon fuel and gas analyses and observed gas temperatures. If the true boiler efficiency be based on the ratio between heat absorbed and heat avail-

able, the efficiency of the 400-hp. boiler, on this calculated basis, would be 67.8 per cent when oil is used as fuel and 58.8 per cent when gas is used as fuel.

If, on the other hand, we accept the calculated figures for heat supplied to the waste-heat boilers as correct and use the observed evaporation as a basis for calculating the boiler efficiency, the efficiency for the 400-hp. boiler would figure out 72.8 per cent when oil is being used and 60.2 per cent when natural gas is used as fuel. The efficiency of the 300-hp. boiler when gas is used as fuel, calculated on this same basis, would be slightly lower than this, or approximately 58.2 per cent.

The tests that have been made by us were not conducted primarily as boiler tests. They were made rather to ascertain the average rate of fuel consumption during the various furnace operations and the corresponding rate of evaporation in the waste-heat boilers.

As might be expected, the rate of evaporation varies from time to time, depending on the condition of the boiler. Following the thorough cleaning of a boiler at the conclusion of a furnace campaign, the boiler efficiency is higher than could be expected at the end of a 90-day campaign. Our observations represent as nearly as possible average operating conditions.

Waste-heat Boiler Practice at the Anaconda Reverberatory Plant

By E. A. Barnard* and George Tryon,† Anaconda, Mont.

(New York Meeting, February, 1934)

The importance of the conservation of the waste heat contained in copper reverberatory furnace gases was realized very early by those in charge of operation at Anaconda. The first attempt to utilize it was in preheating the air used for combustion of the coal. This was done by forcing the air through ducts surrounding the flue-gas downtake and continued down beneath the furnace hearth to the firebox. Any small advantage gained by the heat transferred from the flue gases was more than offset by the losses sustained in cooling the hearth, therefore this process, after several trials, was soon discarded. It was first used at the Old Works as early as 1893 and again for a comparatively short time on the original furnaces at the Washoe smelter, or present works. Water-tube boilers came into general use very rapidly after 1890. Their adaptability for waste-heat steam generation, as well as the saving to be effected by the production of cheap steam from this source, was soon recognized.

The first waste-heat boiler to be used in connection with a matte furnace at Anaconda was installed in September, 1900, at the "Old Lower Works." It consisted of a 100-hp. "Cook," vertical water-tube boiler. This boiler had 65 small perpendicular tubes arranged in concentric circles around a 10-in. tube in the center. These tubes were expanded into 4 ft. 4 in. inside diameter cylindrical steel top and bottom drums which served as the steam and mud drums respectively. This boiler was set in a cylindrical brick housing into which the furnace gases entered just above the mud drum and thence circulated upward around the tubes and steam drum to the stack. This installation proved unsatisfactory, presumably on account of difficulty with dust, and was soon abandoned. However, the possibilities for steam generation were undoubtedly encouraging for shortly afterwards (December, 1900) a 300-hp. Stirling boiler was installed, to be followed by two more in the spring of 1901.

The choice of the Stirling type of boiler for application to copper reverberatory furnaces was particularly fortunate, as the arrangement of the tubes appears to combine to a great extent the maximum contact of the gases with the heating surfaces of the boiler and at the same time

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keeps the resistance to their passage as low as possible. The boiler offers the least possible chance for the accumulation of dust and accretions and is readily accessible for both external and internal cleaning. The fact that this type of boiler has successfully met the changing furnace-

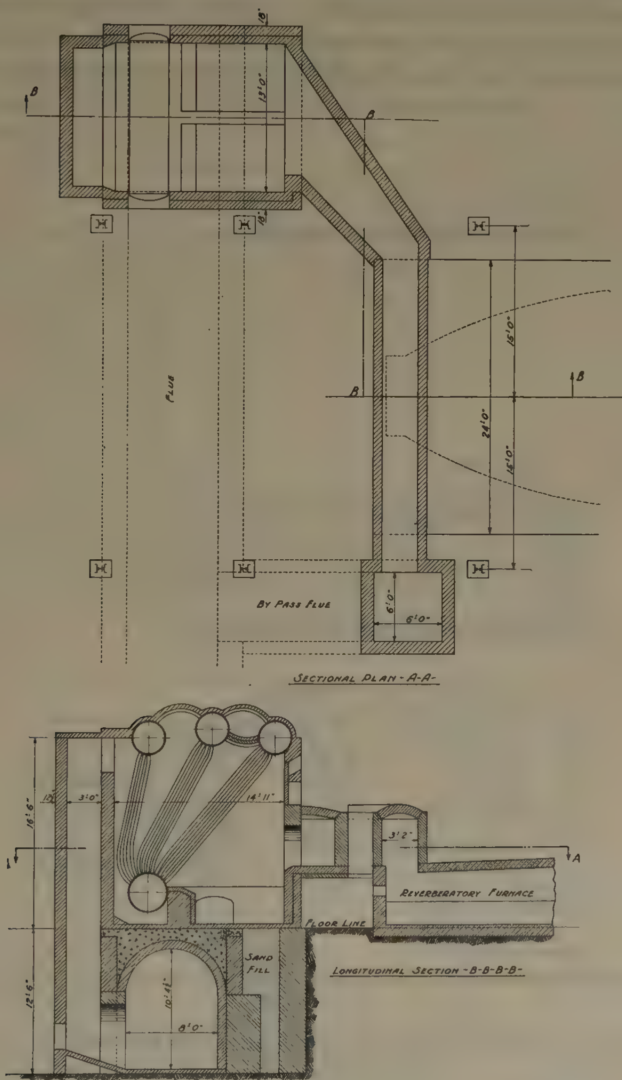

FIG. 1.—ARRANGEMENT OF ORIGINAL FURNACE AND WASTE-HEAT BOILER AT ANACONDA.

operating conditions and has become practically a standard for this class of work speaks well for the foresight of the pioneers who first made the selection.

The original furnaces at the present Anaconda Reduction Works were started without boilers in the early part of 1902 but the air preheat-

ing scheme previously mentioned proving unsatisfactory, the installation of boilers was commenced in June of that year and completed soon thereafter. There were 14 small furnaces arranged in two rows, one row on each side of the reverberatory-furnace building, with a firing aisle between them. Each furnace was connected to a single Stirling boiler, the gases from which passed to an underground flue and thence to a central reverberatory plant stack. The arrangement of furnace and boiler is shown in Fig. 1. Nine furnaces were supplied with 375-hp. and five with

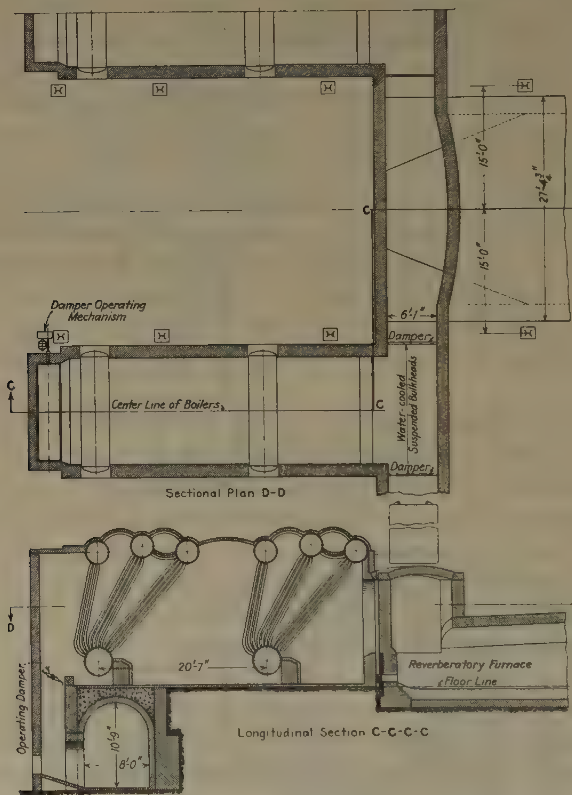

FIG. 2.—ARRANGEMENT OF ENLARGED FURNACE AND WASTE-HEAT BOILERS, ANACONDA.

300-hp. class A Stirling boilers. The furnaces were fired with lump coal burned in fireboxes in which bar grates were used. For a short time forced draft was utilized and baffling was tried on the boilers but by the end of 1902 the baffles had been removed and natural draft had been substituted. The boilers were located between the furnaces in building bays 21 ft. center to center of columns. This fact has had a distinct bearing on the size and arrangement of the boilers used with the larger furnaces developed later.

A new plan designed to collect the gases from all the various smelting operations and deliver them into a common flue system discharging into a 30 by 300-ft. stack on the top of the smelter hill was put into operation in 1903. This gave a greatly improved draft and immediately preceded a program of furnace enlargement. The double row of 14 small furnaces was abandoned and they were replaced by 7 long ones. Another large furnace was added later. The boilers on one side of the building were left in place and those from the other side were moved over and set in tandem with them. As the space between the furnaces was limited, as mentioned above, any arrangement of boilers other than the tandem setting would have necessitated a complete revamping of the furnace building as well as the collecting-flue system. With the advent of pulverized-coal firing the fireboxes were done away with and the furnaces further enlarged, the boilers remaining as they were installed in 1904. Fig. 2 shows the arrangement of boilers to furnace as they were after the furnaces were enlarged and as they have remained to the present time.

In 1915 a large reverberatory furnace was built at the converter plant for handling molten slag. This was equipped with two 860-hp. class M Stirling boilers, only one of which was used at a time.

Table 1 gives a summary of averages of boiler tests covering major operating periods of the reverberatory furnaces. All of these tests were made on the reverberatory waste-heat boilers at the Anaconda Reduction Works (Washoe smelter) except those for period 1, which were taken at the Old Lower Works on the first Stirling boilers. The operating periods were as follows:

Period 1.—Small grate-fired furnaces. Single boilers fully baffled. Charge: hot calcine from gravity concentrates plus secondaries.

Period 2.—Furnaces somewhat larger, boilers not baffled, otherwise about as for period 1.

Period 3.—Furnaces considerably enlarged, grate-fired, with two boilers in tandem. Charge as for No. 1 and No. 2.

Period 4.—Furnaces further enlarged, pulverized-coal fired, boilers tandem. Charge: calcine from bulk flotation and table concentrates, plus secondaries.

Period 5.—About as for No. 4 except calcine from selective flotation concentrates.

Period 6.—Natural gas firing. Charge as for No. 5 plus wet concentrates and both cold and molten converter slag.

Converter Slag Furnace covers operation of a large furnace handling molten converter slag and a coarse highly siliceous charge; 860-hp. Stirling boilers used singly.

TABLE 1.—*Average Boiler Tests Covering Major Operating Periods*

Period No.	1	2	3	4	5	6	Slag Furnace
Date	1901	1902–04	1907	1915–19	1928–29	1933	1919
Location	Old Lower Works Reverbs.	Reverbs.	Reverbs.	Reverbs.	Reverbs.	Reverbs.	Conv. Slag Reverb.
Make of boiler	Stirling	Stirling	Stirling	Stirling	Stirling	Stirling	Stirling
Average boiler rating, hp	300	361.9	696.4	615	750	600	860
Type of boiler installation	Single, fully baffled	Single	Tandem	Tandem	Tandem	Tandem	Single
Character of firing	Grate with run of mine coal	Grate with run of mine coal	Grate with run of mine coal	Pulverized coal	Pulverized coal	Natural gas	Pulverized coal
Coal as fired, tons per 24 hr.	17.31	49.72	60.53	83.86	80.28	(Gas, 2,472,000 cu. ft.)	88.40
Equivalent evaporation from and at 212° F. per 24 hr., lb.	155,059	328,928	423,814	515,181	593,034	749,267	409,003
Average steam pressure gage, lb.	95.6	151.0	157.0	154.8	138.4	124.5	163.4
Average feed-water temperature, deg. F.	50.6	38.6	49.8	41.2	80.3	72.7	41.7
Horsepower developed	187.3	397.2	511.9	622.2	716.2	904.9	494.0
Horsepower, per cent of rating	62.4	109.8	73.5	101.2	95.5	150.8	57.4
Equivalent evaporation from and at 212° F. per pound coal as fired	4.479	3.309	3.501	3.072	3.694		2.313
Equivalent evaporation from and at 212° F. per 1000 cu. ft. gas burned						303.10	
Percentage of total heat of fuel absorbed by boiler. (Based on gross B.t.u.)	37.47	27.67	28.43	24.84	29.52	28.81	19.58
Average boiler inlet temperature, deg. F.					2101	2050	
Average boiler exit temperature, deg. F.			685	830	857.9	810	557.5
B.t.u. in fuel as fired (gross)	11,600	11,600 (Est.)	11,950	12,000	12,140	(Gas, 1021 cu. ft.)	11,466
Percentage moisture in fuel as fired			8.10	2.64	3.10		2.30
Reverberatory data. Hearth dimensions	14′ 4″ × 32′ 1″	19′ × 49′	19′ × 112′	22′ × 133′	23′ 4″ × 134′ 6″	23′ 7″ × 133′ 1″	20′ 10″ × 153′ 0″
Tons solid material smelted per furnace day		142.30	255.70	547.10	577.04	612.27	489.70
Tons solid material smelted per ton coal		2.86	4.22	6.52	7.19		5.54

Steam and Boiler-feed Water

Steam is used primarily for heating and process work, but when there is an excess, as in the summer months, it is used for generating 16, 90 and 900 lb. compressed air. For a short time at the start of the present reduction works a limited amount of electric energy was generated from steam, but this was soon discontinued because of the development of cheap hydroelectric power. The boilers are tested for 175 lb. About 150 lb. pressure is maintained when steam is used for power. No superheaters are employed.

Because the steam consumers are so widely separated, it is practically impossible to return more than a small amount of condensate for boiler-feed purposes. There are over seven miles of interdepartment steam mains in the plant, not including the steam piping within the various buildings themselves. As a consequence the feed is practically all raw water which contains approximately five grains of total hardness per gallon. Most of this consists of carbonates of calcium and magnesium. Small amounts of calcium and sodium sulfate total about one grain per gallon. Some preheating is accomplished by adding exhaust steam from the turbine-driven feed pumps to hot water obtained from roaster-furnace cooling operations. An average feed temperature of about 155° F. is obtained. The water is treated with soda ash and filtered.

The boilers are shut down once each month for cleaning. The scale, which is hard, averages 1/8 to 3/16 in. in thickness and is removed by the use of a turbine. At the same time, dust and accretions are cleaned from the outside of the tubes, boiler settings and connecting flues.

Accretions

The accretions that gather on the tubes result from the mixing of fine dust from the charge with arsenic and other volatile constituents in the gases that condense on the relatively cool boiler surfaces. This forms into a pasty mass, which sticks to the tubes and can only be satisfactorily removed by the use of an air lance. The use of a high percentage of wet charge of late has increased the sticky nature of these deposits, making it necessary to remove them six times per 8-hr. shift, or about twice as often as during the period of pulverized-coal firing and dry charge. The deposits themselves act as an insulator and the incidental infiltration of cold air through open doors due to cleaning tends to lower the boiler efficiency as well as interfere with the furnace draft. The accumulation of accretions undoubtedly contributed to the abandonment of baffling on the boilers. Analysis of two recent samples of deposits taken from boilers operating under present varying furnace conditions are as follows:

Sample	Analysis, Per Cent										
	Cu	As	Sb	Zn	Pb	Bi	S	SiO_2	Fe	Ca	Al_2O_3
1	22.0	11.6	12.0	9.2	3.4	1.4	0.7	9.0	14.2	0.4	1.5
2	19.7	22.0	0.3	3.5	2.3	1.5	0.1	5.7	10.7	0.6	2.7

Tandem Arrangement of Boilers

As mentioned previously, this arrangement was necessitated by limitations of building space, but considering the large increase in furnace sizes and capacities, it has worked out comparatively well from a boiler standpoint. Tests indicate that on the tandem boilers about 80 per cent of the steam is generated on the first boiler. The tandem arrangement, together with the small size of the boilers, results in a high gas velocity and a considerable loss of draft through the boilers. Draft readings approximate as shown in Table 2. For the sake of comparison, tests

Table 2.—*Draft Readings*

	Draft Preceding Boilers, In. H_2O	Draft Following Boilers, In. H_2O	Draft Loss through Boilers, In. H_2O
Boilers in Tandem			
	0.29	0.81	0.52
Slag-furnace Boilers			
Both boilers operating	0.16	1.10	0.94
One boiler operating	0.16	1.08	0.92

were run on the slag-furnace boilers, which are arranged so that one 860-hp. unit may be run singly or two may be run in parallel on one furnace. One boiler running singly made 77 per cent as much steam as the two running in parallel with furnace conditions the same.

Summary

In conclusion, this paper shows that the boiler efficiencies based on the fuel burned in the reverberatories have remained nearly constant regardless of the kind or amount of fuel burned. An exception to this is period 1, in which the boilers were baffled. Period 2 on single boilers shows an efficiency about the same as the later periods with tandem boilers.

The boilers are small and placed in tandem because of building construction limits. The high cost of reconstructing the plant for higher boiler efficiency would not be warranted.

The small size of the boilers, together with the accretion problem, has necessitated the elimination of all boiler baffles.

In general, the boilers have continued to give satisfactory furnace conditions through a wide range of furnace capacities.

Waste-heat Boiler Practice of Nevada Consolidated Copper Corporation

By N. W. Sager* and H. W. Mossman,† McGill, Nevada

(New York Meeting, February, 1934)

The arrangement and general dimensions of the reverberatory furnaces and waste-heat boilers for the Nos. 2 and 3 smelting units at the McGill plant of the Nevada Consolidated Copper Corporation are shown in Fig. 1. Each of the two reverberatory furnaces has its own set of waste-heat boilers, and before February, 1932, each furnace, when it was in operation, delivered waste-heat gases to its own boilers only. In January, 1933, the idea was suggested of using some of the waste-heat boilers as economizers to operate in conjunction with the waste-heat boilers that were generating steam. The execution of this idea gave rise to the gas-flow scheme depicted in Fig. 1, the advantages of which were obvious, particularly so in view of the fact that for some months prior to the adoption of this scheme the smelting had been confined to No. 2 furnace, with No. 3 furnace and its boilers idle. The waste-heat gases from No. 2 furnace pass through the boilers serving this furnace as steam-generating units, thence through the original exit duct to a point where a bulkhead diverts the gases in a reverse direction through two of the boilers of No. 3 furnace. After the gases have traversed these two boilers, another bulkhead placed in the duct between No. 3 furnace and its boilers causes them to traverse still one more boiler, from which the gas path leads to the stack. In this scheme the last three boilers traversed by the gases, i.e., the boilers behind No. 3 furnace, function as economizers and preheat the feed water for the two steam-generating units directly serving No. 2 furnace.

This discussion will give consideration to the results obtained in waste-heat recovery by the scheme shown in Fig. 1, and, further, will compare these results with those obtained before the boilers of No. 3 furnace were set up to act as economizers, but before going into the detail of heat generation, distribution and recovery, will present some additional details of the equipment.

All boilers in this waste-heat plant are of the Stirling type. The two behind No. 2 furnace, acting as steam-generating units, of class 13, are 40 tubes wide and rated at 1112 boiler hp., while the three boilers behind No. 3 furnace, acting as economizers, are also of class 13, but 28 tubes

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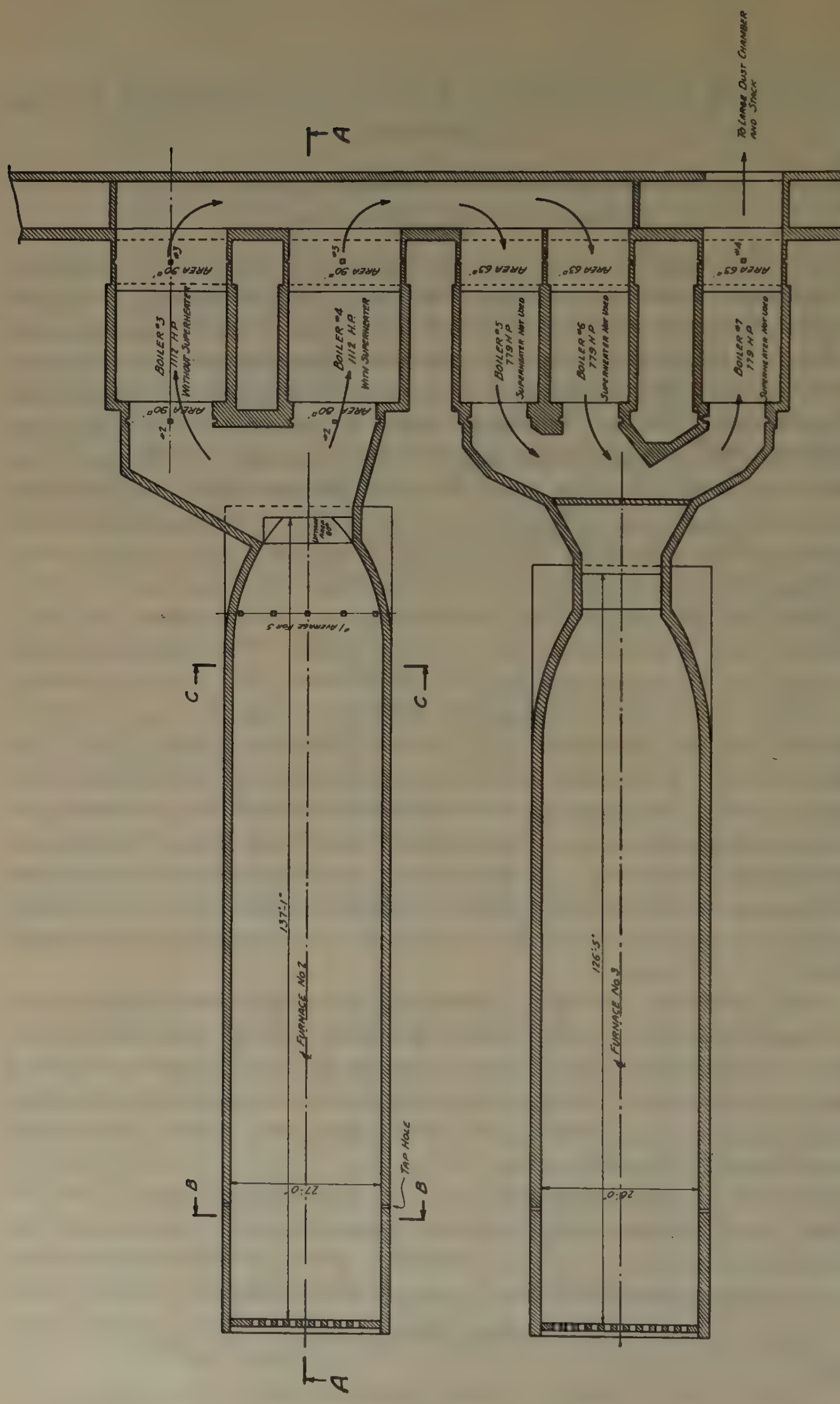

FIG. 1.—FLOW OF REVERBERATORY GASES THROUGH WASTE-HEAT BOILERS, NEVADA CONSOLIDATED COPPER CO., STEPTOE PLANT.

wide and rated at 779 boiler hp. The boilers are equipped with "valve in head" soot blowers; those of the boilers generating steam using steam for soot-blower operation while those of the economizer boilers utilize air for soot-blower operation. Hand-operated air lances are used to supplement the soot blowers, particularly in the front tube banks of the boilers directly behind No. 2 furnace, which collect fairly heavy deposits of dust from the furnace.

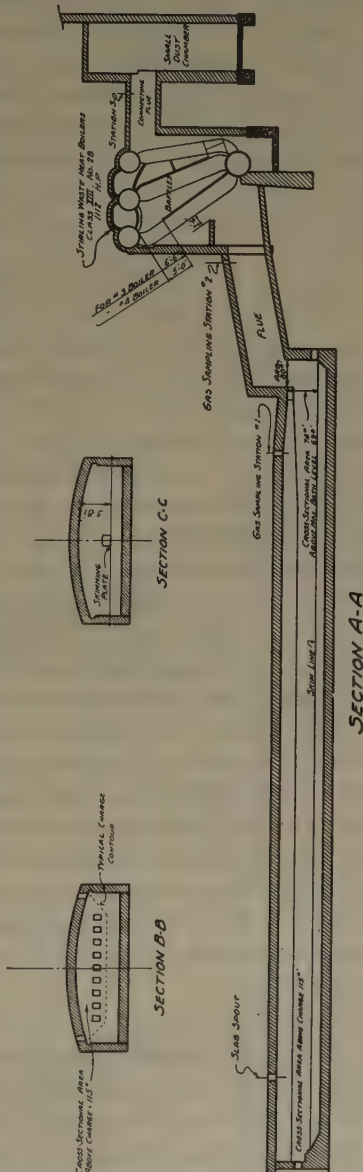

FIG. 1.—(Continued.)

Steam generation from the waste-heat boilers is measured by electrically operated flowmeters, each boiler having its own meter. These meters measure the net steam delivered to the power plant, that being required for soot-blower operation not being credited as boiler evaporation. These meters have been checked, at suitable intervals, against a master waste-heat steam-meter, and also against a V-notch meter which can be arranged to measure the feed water going to the boilers. Under these conditions the waste-heat evaporation is accurately known.

The feed water supplied to these boilers is approximately 85 per cent condensate, the remainder being a softened water, treated by lime and sodium aluminate, so that the treated water has a hardness of 1.2 grains per gallon. The feed water is heated by exhaust steam in an open air heater at the power plant, the water temperature at the heater ranging from 190° to 200° F. From here the water passes under pressure through a water screen in a separately fired superheater, where the temperature is raised 5° to 7° higher, and thence to the waste-heat boiler plant. However, several degrees of temperature are lost between these last two points, so that the final temperature of the feed water entering the waste-heat plant is in the range of 190° to 200° F.

Under the present scheme of waste-heat boiler operation, the feed water passes progressively through No. 7, No. 6 and No. 5 economizer boilers and thence to the steam-generating boilers 3 and 4. The temperature of the feed water is raised approximately 120° in passing through the three economizer boilers.

Steam in the waste-heat boilers is generated under a pressure of approximately 190 lb. gage, that from No. 4 boiler being superheated to approximately 500° F., while the steam from No. 3 boiler is delivered at the saturated temperature. The temperature of the combined steam from the two units is in the neighborhood of 450° F.

Reverberatory-furnace Practice

Pulverized coal is used as a fuel in the McGill furnaces, the coal being prepared in a separate building where grinding is performed in a Hardinge ball mill. The pulverized fuel is delivered as a coal-air mixture (approximately 10,000 cu. ft. per minute), at about 6 in. water-gage head, to the furnace burners. No. 2 furnace has nine 6-in. diameter pipe burners. A quantity of booster air (approximately 2000 cu. ft. per min. at ½ lb. pressure), in addition to that required for conveying the fuel, is admitted through pipes that extend part way into the burners. This air has an aspirating effect on the coal-air mixture entering the burners and also assists in projecting the fuel-air mixture into the furnace with a fairly high velocity. The velocity of this stream has the further effect of assisting in the aspiration of the required quantity of secondary air through the openings surrounding the burners.

The present metallurgical practice in the McGill reverberatories, locally described as "direct smelting," involves the smelting of a wet, raw charge, rather than the smelting of hot or dry calcines. The raw charge to the furnace has a moisture content of approximately 9 per cent, the temperature of this material being that of the ambient air. The matte grade produced is approximately 35 per cent copper. Charging of the raw material into the furnace is accomplished by introducing the material through holes in the furnace roof adjacent to the side walls.

Fuel

Under the present average rates of operation the daily coal requirements for the No. 2 furnace at McGill are roughly 117 tons. This coal is a bituminous grade from the Utah field, received at the plant in the forms of slack and dust. A typical average proximate analysis of this coal on the as-received basis follows: moisture, 6.82 per cent; volatile matter, 31.35; fixed carbon, 55.27; ash, 6.56; B.t.u., 12,140.

Some moisture elimination is accomplished before this coal goes to the pulverizer but under the present arrangement the quantity of moisture entering the pulverizer is the same as that entering the furnace burners,

since no moisture elimination is effected between the pulverizer and the burner. A portion of the moisture, however, may appear associated with the primary air rather than with the coal in the coal-air mixture.

In drying and handling coal there are certain physical losses. From data available it is estimated that this loss, in addition to moisture elimination, is approximately 2 per cent of the coal weight as received. A typical ultimate analysis of this coal on the as-received basis and on the basis of delivery to the burners is given in Table 1. A typical screen analysis of the pulverized coal is shown in Table 2.

TABLE 1.—*Analysis of Utah Coal Used at McGill*

Ultimate Analysis	As Received, Per Cent	In Burners, Per Cent
H_2O	6.82	4.33
Carbon	66.42	68.19
Hydrogen	5.06	5.19
Oxygen	12.53	12.88
Nitrogen	1.42	1.45
Sulfur	1.61	1.66
Ash	6.14	6.30
Total	100.00	100.00

TABLE 2.—*Typical Screen Analysis of Coal Used*

Mesh	Individual Weight, Per Cent	Cumulative Weight, Per Cent
+ 48	0.7	
+ 65	1.0	1.7
+100	4.7	6.4
+150	6.5	12.9
+200	9.2	22.1
−200	77.9	100.0

TABLE 3.—*Combustion Products*[a]

Constituent	Gas, Lb.	Cu. Ft. at Standard Conditions
CO_2	2.499	20.25
N_2	6.986	89.10
H_2O	0.467	9.31
SO_2	0.033	0.18
	9.985	118.84

[a] Per pound of coal with no excess air.

Calculating the air requirements for perfect combustion of this coal on the as-received basis shows that 8.923 lb. of air are required per pound

of coal, while on the basis of coal delivered to the burner this would be 9.132 lb. of air per pound of coal. On this latter basis the combustion products per pound of coal with no excess air would be as shown in Table 3.

Gases of Combustion

The location of the various stations for making gas analyses and temperature measurements is shown on Fig. 1. Station 1 includes five sampling points across the reverberatory section. Results secured from the central point are compared in Table 4 with those secured from a five-point traverse across this section. Single sampling points are indicated for each of the steam-generating boilers (Nos. 3 and 4) on the inlet and exit gas sides, while the final gas condition is determined at station 4, located at the gas outlet from the last economizer boiler (No. 7). For operating conditions prior to the use of boilers 5, 6 and 7 as economizer boilers, the final gas condition was that as obtained at station 3 on the exit side of boilers 3 and 4.

Table 4.—*Representative Temperatures and Gas Analyses at the Various Stations*

Station	Temperature Deg. F.
1-a	2000
1-b	2000
2	1800
3	664
4	323

Orsat Gas Analyses (Volumetric), Per Cent

	Station 1-a (Center)	Station 1-b (Composite)	Station 2	Station 3	Station 4
CO_2 and SO_2	16.1	10.5	9.5	9.0	7.9
CO	0.6				
O_2	2.6	9.4	10.5	11.1	12.3
N_2	80.7	80.1	80.0	79.9	79.8
	100.0	100.0	100.0	100.0	100.0

These Orsat analyses for station 1-b when corrected for H_2O and SO_2 are as follows:

CO_2	9.7	O_2	6.0	H_2O	7.5
CO	0.0	N_2	75.0	SO_2	1.8

Steam Production

Waste-heat steam production and the fraction of heat recovery indicated thereby will be shown for two different periods. The first period includes results obtained during a 15-day run in January, 1933,

just previous to the adoption of the economizer boiler scheme. The second period includes results obtained during a 30-day run in February and March of 1933.

The essential data pertaining to steam generation and heat recovery are:

B.t.u. per pound coal as received	12,140
B.t.u. per pound coal as burned	12,460
Feed-water temperature, deg. F	193
Feed-water temperature leaving economizers (when operating), deg. F	310
Steam temperature, deg. F	447
Superheat, deg. F	63.9
Gage pressure, lb. per sq. in	191
Absolute pressure, lb. per sq. in	203
Total heat in steam, B.t.u. per lb	1,237.9
Total heat in feed water, B.t.u. per lb	161.0
Heat input to steam, B.t.u. per lb	1,076.9

	Without Economizer Boilers 15 Days Operation in Jan., 1933	With Economizer Boilers 29.7 Days Operation in Feb. and Mar., 1933
Steam produced per pound coal as received, lb.	6.18	6.84
Steam from and at 212° per pound coal as received, lb	6.91	7.65
Steam produced per pound coal as fired, lb	6.33	7.02
Steam from and at 212° per pound coal as fired, lb	7.08	7.85
Per cent recovered in steam of B.t.u. in coal as received	54.8	60.7
Per cent recovered in steam of B.t.u. in coal as burned	55.9	61.9

Supplementary Remarks

In order to gain a better insight as to the why and wherefor of differences in waste-heat steam recovery from reverberatory-furnace exit gases it is necessary to consider the multiplicity of factors that in one way or another affect the quantity of heat recoverable in the form of steam.

As one means of establishing the magnitude of the various factors influencing the fraction of waste heat recovered it might be suggested that a heat balance be developed for the reverberatory-furnace waste-heat boiler system. Actually when an attempt is made to do this with the typical data set forth in this report, and similarly with the Clarkdale data (see page 246), it is found that the amount of heat leaving in the steam plus that leaving in the exit gases from the boilers is greater than the amount of heat entering the waste-heat boilers as calculated from the

typical uptake-gas analyses and temperatures. At once, of course, the statement is justified that some or all of the available data are incorrect. At McGill, the available means of steam quantity measurement are such that this factor may be set down as being accurately known. The remaining sources of error are then traceable to gas analyses and gas temperatures, wherein the possibility of error rests not so much with the accuracy of a single measurement as it does in the failure to establish a true mean set of values, such as would result from a complete traverse of the gas-flow path at the various stations. The truth of this statement is well shown in the Orsat analyses (Table 4) where the first two columns contrast the gas analyses at a single point in the center of the gas path with the average gas analysis for five points on this section. A heat balance based on the gas sample at the center of the furnace fails entirely to account for the mass of gas required to convey the quantity of heat needed to balance the steam output and that contained in the final exit gases. A closer approach to a heat balance agreement is obtained when the composite gas analysis of the second column is made the basis of calculation.

Gas analyses similar to that of the first column have long been reported as representative of the uptake gases in the McGill reverberatory furnaces, although at the same time it has been known that such analyses with the quantity of waste-heat steam produced gave heat quantities that were in disagreement. Apparently the only way to reconcile these two factors was to account for more gas than was generally shown by the analyses. Not only is this requirement dictated by theory; it is actually substantiated by recent field tests at McGill.

In comparing in a more or less rough manner the waste-heat recovery from various installations, without resorting to the details of a complete heat balance, it appears desirable that some weight should be given to the influence that sulfur combustion has on the final results. Thus for the February-March period covered in this report, sulfur was eliminated at such a rate that for each pound of coal burned 0.302 lb. of sulfur was driven off from the charge. If it be assumed that this sulfur was burned to sulfur dioxide and its calorific value taken as 4050 B.t.u. per pound, then for each 12,140 B.t.u. liberated by the coal there were also liberated 1220 B.t.u. from the sulfur. This is roughly 10 per cent of the heat value of the fuel, so that its effect on the final result is appreciable.

As has been mentioned, "direct smelting" is the present practice at McGill. In view of the fact that the Clarkdale data are based on calcine smelting, it may be well to show some results obtained at McGill during the first 10 months of 1932, when calcine smelting was being done. During this period an average of 121.4 (as received) tons of coal per furnace day was being fired, and an average of 5.66 lb. of water was actually evaporated in the waste-heat boilers per pound of reverberatory

coal burned. Feed water and steam conditions were approximately the same then as shown earlier in this paper and if it is assumed that the calorific value of the coal was the same as that herein reported the heat recovered in the waste-heat steam was 50.2 per cent of that in the fuel. This compares with 54.8 per cent with direct smelting but without the economizer boilers, and with 60.7 per cent with direct smelting and the economizer boilers.

Waste-heat Boiler Practice at the United Verde Copper Company Smelter

By J. R. Marston,* Clarkdale, Arizona

(New York Meeting, February, 1934)

The arrangement of the larger furnace at the United Verde smelter, together with the boiler layout and connecting flues, is shown in Fig. 1. The important dimensions are given in the figure, and cross-sectional areas of combustion space, flues, etc., are recorded. The location and extent of the baffling in the boilers also are shown because of their importance in the interpretation of the data.

Two boilers serve each reverberatory furnace. They are of the class M-26 Stirling type (rated at 713 boiler hp. each). Steam is generated at 178-lb. gage pressure at a total temperature of 528° F., from feed water at a temperature of 113° F. The feed water is condensate from turbogenerators plus make-up softened by the lime-soda process. The make-up is rather large in amount as it must match the amount of steam consumed in reciprocating blowing engines and the like, which are not served by surface condensers. As a result the boiler feed water has a hardness of 1.5 to 3.0 grains per gallon, but this results in the formation, during six months to one year of boiler operation, of scale only of "egg-shell" thickness or less.

The boilers are set at the same elevation as the tapping floor of the furnace, which results in the introduction of the waste-heat gases at the top of the front bank of tubes and necessitates the inversion of the usual baffling arrangements.

The boilers are equipped with valve-in-the-head soot blowers, which keep most of the dust blown off the surface of the tubes, but it is necessary to supplement the soot blowers by high-pressure air lances, inserted by an attendant through side doors in the boiler setting.

Deposits of coal ash and of dust from the furnace charge accumulate in the flues between the reverberatory furnaces and the boilers and this necessitates cleaning of these connecting flues by hand during the greater part of one of the three 8-hr. shifts during each 24 hr. This cleaning operation requires the opening of clean-out doors, which, of course, has some effect on the yield of steam.

Reverberatory Furnace Practice

The United Verde furnaces are fired with pulverized coal of sub-bituminous type of which the screen and chemical analyses are given in Table 1.

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* Research Chemist, United Verde Copper Co.

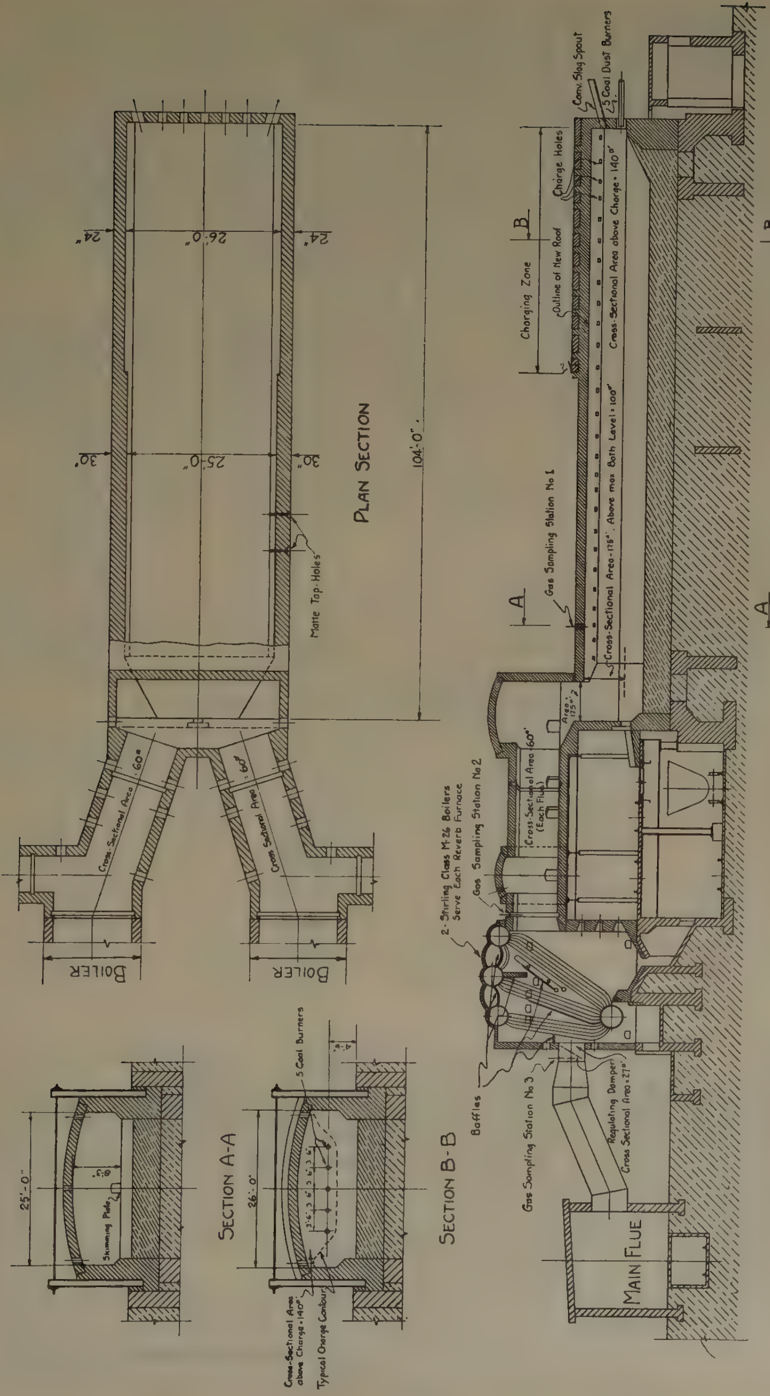

FIG. 1.—SECTION THROUGH FURNACE, WASTE-HEAT BOILER AND FLUES, UNITED VERDE SMELTER.

The furnaces are used for the production of matte containing 20 to 30 per cent copper from the smelting of a solid charge containing 95 per cent hot calcine, together with the treatment of molten converter slag, returned from the converting of the matte. The temperature of the calcine is approximately 800° F. As a rule there is no limerock added to the charge, therefore there is no gas originating from the decomposition of carbonates. There is a small amount of SO_2 produced through elimination of sulfur from the solid charge, which is added to the gas volume produced by the combustion of the coal. This small amount of additional SO_2 is recorded in the data listed in Table 2. The solid charge is introduced along the side walls through holes in the roof of the furnace. About 90 per cent of the solid charge is introduced into the zone 35 ft. long beginning at the burner end of the furnace.

TABLE 1.—*Coal Analysis*

Chemical Analysis		
	As Received, Per Cent	As Delivered to Burners, Per Cent
H_2O	11.2	5.20
C	60.0	64.10
H_2	4.7	5.00
O_2	11.5	12.27
N_2	1.1	1.17
S	1.0	1.06
Ash	10.5	11.20
	100.0	100.00
B.t.u. per pound	10790	11520

Screen Analysis of Pulverized Coal		
Mesh	Individual, Per Cent	Cumulative, Per Cent
On 65	6.9	6.9
100	7.9	14.8
150	13.3	28.1
200	12.2	40.3
300	51.6	91.9
350	3.0	94.9
Through 350	5.1	100.0
	100.0	

Fuel.—One reverberatory furnace of the size shown normally burns 112.58 tons of pulverized coal in 24 hr. The fuel is received at the smelter as screened slack from Gallup, N. M. It is sub-bituminous in type and has the following approximate analysis: moisture, 11.20 per cent; volatile

matter, 36.60; fixed carbon, 41.70; ash, 10.50. As the moisture is nearly all combined moisture, the coal is dried until the moisture content is 5.2 per cent, then pulverized and delivered to the furnace bins. The ultimate analyses of the mine-run and pulverized products are given in Table 1.

As a result of the reduction in moisture content to 5.2 per cent H_2O, the coal as burned is 93.7 per cent of the weight of the coal received. Of course there are physical losses of coal in the drying and pulverizing process other than the mere loss of weight due to the elimination of moisture. The data in this paper refer to the coal as burned; that is, the pulverized coal containing 5.2 per cent moisture, which actually passes through the burners and into the furnace. One pound of pulverized coal requires 8.6123 lb. of air for combustion and the perfect combustion results in the products shown in Table 2.

TABLE 2.—*Products of Combustion of Pulverized Coal*

	Gas, Lb.	Gas, Cu. Ft. at Standard Conditions
CO_2	2.3503	19.044
N_2	6.6268	84.644
H_2O	0.5020	10.000
SO_2	0.0212	0.119
	9.5003	113.807

TABLE 3.—*Gas Temperatures and Analyses*

Gas-sampling Station	Temperature, Deg. F.
1	2200
2	1900
3	600

GAS ANALYSES (VOLUMETRIC), PER CENT

	Station 1		Station 2		Station 3	
	As Measured by Orsat Apparatus	As Corrected for H_2O and SO_2	As Measured by Orsat Apparatus	As Corrected for H_2O and SO_2	As Measured by Orsat Apparatus	As Corrected for H_2O and SO_2
CO_2	18.0	16.2	17.8	16.0	17.0	15.3
CO	0.3	0.3				
O_2			0.5	0.5	1.5	1.3
N_2	81.7	73.0	81.7	73.2	81.5	73.5
H_2O		9.4		9.2		8.9
SO_2		1.1		1.1		1.0
Total		100.0		100.0		100.0

Gases of Combustion.—On Fig. 1 the locations of three gas-sampling stations are marked: station No. 1 being through the roof of the reverberatory furnace close to the gas uptake; station No. 2, at the entrance to the boilers and station No. 3 at the exit of the boilers. Typical gas analyses and temperatures at these stations are given in Table 3.

Steam Production

As mentioned, boiler campaigns are from six months to one year, with an average of about eight months, so for the purposes of reporting the yield of steam, actually measured quantities of water and fuel have been summed up for an eight months' period. The data obtained may be summarized as follows:

Pulverized coal as received contained		10,790 B.t.u. per pound[a]
Pulverized coal as burned contained		11,520 B.t.u. per pound[a]
Feed-water temperature	=	113° F.
Temperature of steam	=	528° F.
Superheat of steam	=	150° F.
Gage pressure	=	178 lb. per sq. in.
Absolute pressure	=	190 lb. per sq. in.
Total heat in steam	=	1,283.8 B.t.u. per lb.
Total heat in feed water	=	81.0 B.t.u. per lb.
Heat input to steam	=	1,202.8 B.t.u. per lb.

Steam generated:

Per pound wet slack coal (coal-mine weights)	3.75
Per pound wet slack coal from and at 212° F.	4.65
Per pound pulverized coal actually burned	4.18
Per pound pulverized coal from and at 212° F.	5.18
Percentage recovered in steam of B.t.u. in coal actually paid for	41.9
Percentage recovered in steam of B.t.u. in coal actually burned	43.6

[a] Calorimeter determination, H_2O condensed.

Summary

Tons solid charge smelted per 24 hr.	883
Tons pulverized coal (5.2 per cent H_2O) burned per 24 hr.	112.58
British thermal units per ton solid charge	2,938,000
Pounds water evaporated per ton solid charge smelted	1,067
Pounds water evaporated per million B.t.u	363

Efficiency based on pulverized coal actually burned at the furnace:

	Per Cent
To smelting, conduction, convection and radiation losses, etc.	43.0
Transferred to steam	43.6
Leaving in stack gases	13.4

Waste-heat Boiler Practice at Miami

By P. D. I. Honeyman* and P. A. Faust,† Inspiration, Arizona

(New York Meeting, February, 1934)

At the Miami plant of the International Smelting Co., Inspiration, Ariz., there are four reverberatory furnaces, all 120 ft. long. At present only one of these furnaces is in operation. It is operated as a wet-charge furnace, the charge being distributed for about 65 ft. down the furnace length. Usually this charge contains about 12 per cent moisture and runs 25 per cent sulfur, of which there is eliminated in the furnace from 25 to 30 per cent.

The furnace is oil-fired, the fuel being burned in a battery of Cananea-type, low-pressure burners. The flame is kept short and conditions so adjusted as to obtain complete combustion within the furnace. The average furnace draft, taken through the verb arch, amounts to only 0.02 in. of H_2O. Draft is purposely kept low to cut down infiltration through the flues and boiler settings, and control dampers are regulated to maintain this uniform condition throughout the charging as well as the smelting period.

Hot gases discharge from the furnace uptake through a common header flue to the waste-heat boilers. The general layout and dimensions of the plant are shown in Fig. 1. Four Stirling boilers, two of class M-26 (713 hp.) and two class M-31 (850 hp.), serve the furnace. Three usually are sufficient to take care of normal operation, the fourth boiler being used as a spare when necessary.

The distribution of hot gases to the various boilers is regulated by the dampers in the boiler exit flues. Normally the boilers closest to the furnace uptake would evaporate more water than do those farther away, but dampers are regulated so as to distribute fairly well the load between the several boilers.

Steam from the waste-heat plant is delivered to the adjoining power house of the Inspiration company, where it is used both in the Nordberg blowing engines and for electrical generation. Steam pressures and temperatures recorded are those in the main power-plant header and are thus inclusive of all losses in about 500 ft. of 10-in. line. Steam for smelter heating and auxiliary use is taken from the main waste-heat header, is reduced to 125 lb. and distributed through separate lines.

Boiler feed water is supplied by the power plant and consists of returned condensate, augmented to a large extent by make-up water

* Superintendent, International Smelting Co.

† Master Mechanic, International Smelting Co.

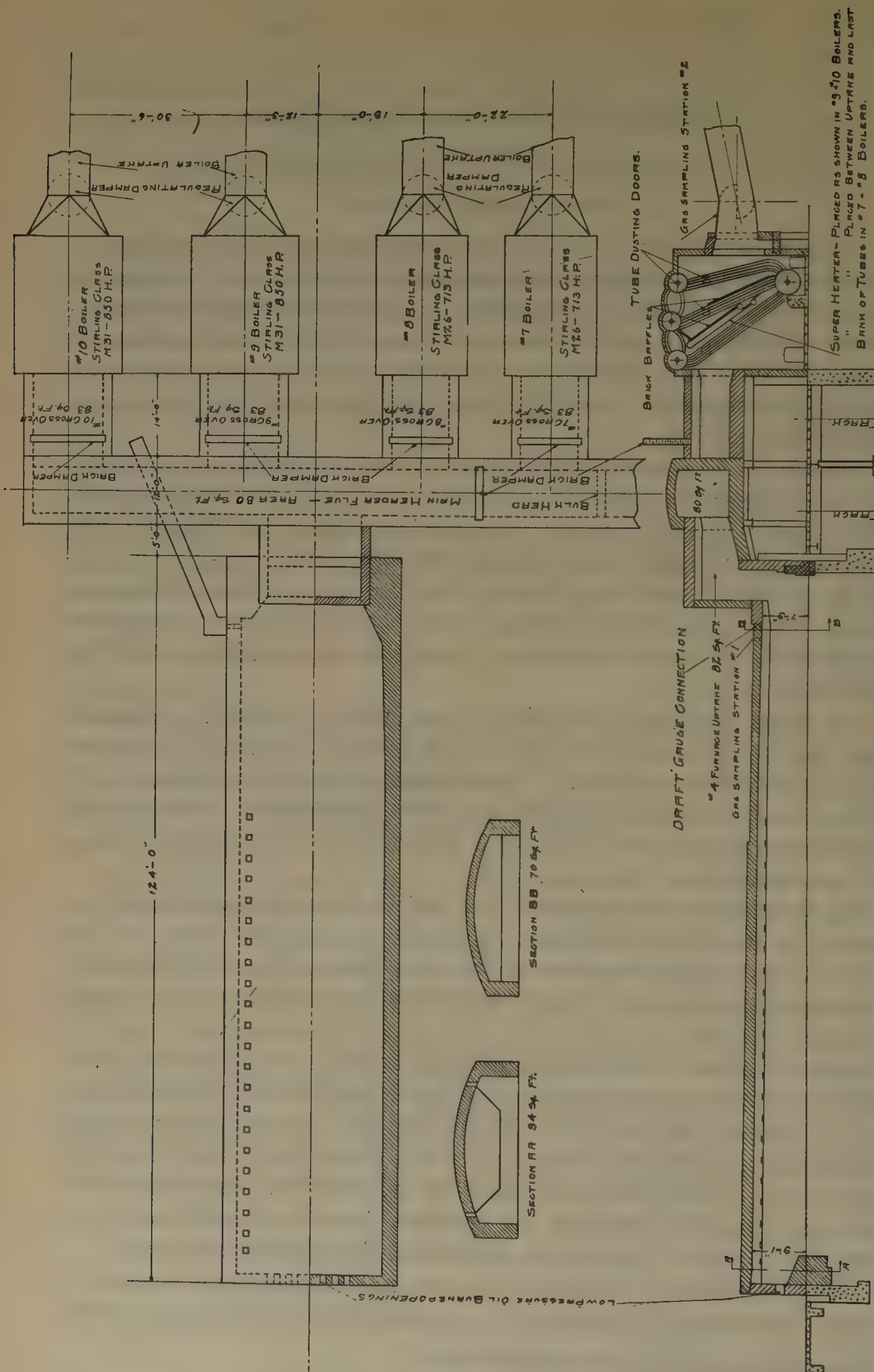

FIG. 1.—ARRANGEMENT OF NO. 4 REVERBERATORY FURNACE FLUES AND WASTE-HEAT BOILERS, INTERNATIONAL SMELTING CO.

that has been treated by the lime-soda ash process. All feed water is measured to the boilers through a tipping weighmeter and a Lea V-notch recorder. Boilers are blown down about once a shift, all such blowoff being also metered.

A certain slight amount of scale forms in the boilers and it is necessary from time to time to scale the tubes, particularly those of the front banks.

There are no soot blowers but boilers are dusted once a day with a hand lance using high-pressure steam. The accumulation of dust within the boiler is comparatively light and settings do not have to be cleaned out more than once in several months.

The boilers are on the same floor level as the reverberatory furnace and since the hot gases enter at the top it is necessary that the baffles be installed in a reverse manner to the baffling usually found in a direct-fired boiler.

The larger, more recently installed M-31 class boilers have superheaters just back of the first bank of tubes, as shown in Fig. 1. In the older M-26 boilers the superheaters are behind the last bank of tubes, just in front of the boiler uptakes. In the older boilers, when wet charging was adopted extra baffling was installed in front and behind the superheaters to further assist in heat recovery, and the production of steam of better quality from these boilers.

Waste-heat recovery based on heat input to the steam as delivered to the power plant, and corrected for blowoff, amounts to 41.3 per cent of the original heat content of the fuel burned in the reverberatory furnace. This amounts to an equivalent unit evaporation (from and at 212° F.) of 7.75 lb. of water per pound of oil burned.

A further way of expressing this result in terms of heat units per ton of charge smelted is as follows:

	Millions B.t.u. per Ton of Charge
Gross fuel input	5.597
Recovered in steam	2.310
Net furnace consumption	3.287

Fuel

Fuel oil is received in tank cars and unloaded directly to the smelter unit tanks or to storage, depending on local conditions. Pertinent facts concerning fuel oil are as follows:

Average daily fuel consumption, bbl	450
Specific gravity, deg. Bé	13–14
Weight per barrel, lb	340.6
Heat content per pound, B.t.u	18,220
Heat content per barrel, B.t.u	6,205,732

Fuel oil is heated to 120° F. for pumping, is delivered to the furnace at 70 lb. pressure, where it is metered and further heated to from 190° to 200° F. before being delivered to the burners. To insure regular flow, oil temperatures are controlled all along the line by thermostats.

TABLE 1.—*Gas Data*

Furnace uptake temperature, deg. F	2300		
Average boiler exit gas temperature, deg. F	500–550		
Draft at verb arch, in. H_2O	0.02		
	$CO_2 + SO_2$	O_2	CO
Analyses, per cent			
At verb arch	16.0	0.5	
Behind boilers	11.7	6.8	

TABLE 2.—*Data on Steam*

Feed-water temperature, deg. F	168.4
Steam pressure gage, lb. per sq. in	184.8
Steam pressure, abs., lb. per sq. in	197.7
Steam temperature, deg. F	413.1
Superheat, deg. F	32.2
Total heat in steam, B.t.u	1,218.5
Heat in feed water, B.t.u	136.4
Heat input to steam, B.t.u	1,082.1
Actual unit evaporation per pound oil burned, lb	6.95
Equivalent unit evaporation (from and at 212° F.) per pound oil burned, lb	7.75
Heat recovered in gross steam produced of heat in fuel burned, per cent	41.3
Heat input of fuel consumed per ton of charge smelted, B.t.u	5,597,570
Water evaporated per ton of charge smelted, lb	2,135
Water evaporated per million B.t.u. of heat input, lb	382

Waste-heat Boiler Practice at the Magma Copper Company Smelter

By J. H. Rose,* Superior, Arizona

(New York Meeting, February, 1934)

The arrangement of the reverberatory furnace and the two waste-heat boilers at the Magma Copper Company's smelter at Superior, Ariz., is shown in Fig. 1. The inside dimensions of the furnace at the slag line

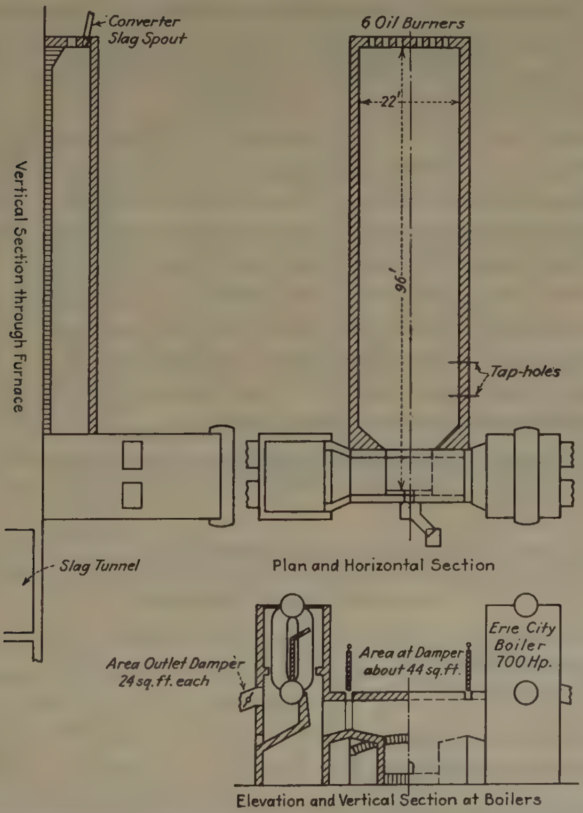

Fig. 1.—Arrangement of reverberatory furnace and waste-heat boilers, Magma Copper Co.

are: length, 96 ft.; width, 22 ft. The boilers are Erie City vertical type with a rated capacity of 700 hp. The 3-in. boiler tubes are arranged in two banks with a vertical baffle between. The banks are 30 tubes wide by 9 tubes deep—a total of 540 tubes.

* Smelter Superintendent, Magma Copper Co.

Steam is generated at 200-lb. gage pressure and a total temperature of 500° F. from feed water at a temperature of 250° F. The feed water is condensate from turbines plus make-up softened by a Zeolite softener. The water is heated by two-stage extraction heaters using partially expanded steam bled from the turbine.

Originally the boilers were equipped with soot blowers but these were not effective in dislodging the heavy dust and are not now in use. Tubes are kept clean by hand lances using compressed air; requiring one man per shift to look after two boilers. The greatest dust attachment occurs in the front row of tubes, where the hot gases strike the tubes at high velocity. The dust particles are in a semiliquid condition and build up rapidly on the tubes unless dislodged at frequent intervals.

Reverberatory Furnace Practice

The furnace is fired with fuel oil of the heavy type known as Bunker C. The oil is heated to 200° F. and supplied at 100 lb. gage pressure to six burners using air at a pressure slightly under 2 lb. gage pressure.

The furnace charge consists of 80 per cent hot calcine at a temperature of 800° F. and the remainder of cold material consisting of crushed ore, converter byproducts, Cottrell dust and other flue dust. The matte averages about 40 per cent copper. The daily tonnage of solid charge has varied from 300 to 700 tons, according to the supply available, and the oil consumption varies from 180 to 350 bbl. per day. The furnace is side-charged for a distance of 50 ft. from the bridge wall, and the greater part of the smelting is done in the first 25 feet.

Samples of gases of combustion, taken at the entrance to the boilers, analyzed by Orsat apparatus show: $CO_2 + SO_2$, 14.0 per cent; CO, 0.0; O_2, 1.0; N_2, 85.0.

Data on steam production are given in Table 1.

Table 1.—*Steam Production*

Estimated heat value of oil, B.t.u. per lb		18,500
Feed-water temperature, deg. F		250
Steam temperature, deg. F		500
Gage pressure, lb		200
Total heat in steam, B.t.u. per lb	1,265	
Total heat in feed water, B.t.u. per lb	218	
Heat input to steam, B.t.u. per lb	1,047	
Water fed to boilers per pound oil burned, lb		6.30
Equivalent evaporation from and at 212°, lb		6.80
Recovered in steam of B.t.u. in oil burned, per cent		35.6
Temperature of boiler exit gas, deg. F		645

Waste-heat Boiler Practice at the Garfield Smelter

By R. A. Marriott,* Garfield, Utah

(New York Meeting, February, 1934)

The steam-generating plant at the Garfield smelter of The American Smelting and Refining Co. consists of two divisions in which the boilers are classed as direct-fired or waste-heat. The direct-fired boiler plant consists of eight 330-boiler hp. type K-19 and S-17 Stirling boilers located in the boiler room adjacent to the power house, and two 330-boiler hp. Stirling type K-19 boilers in the reverberatory building, which formerly were connected to a reverberatory furnace but now are arranged for direct firing. All direct-fired boilers are now fired with natural gas.

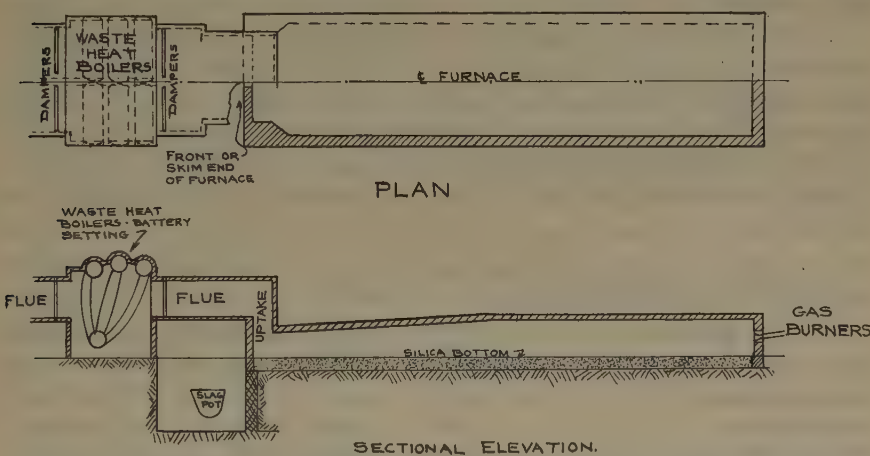

Fig. 1.—Arrangement of reverberatory furnace and waste-heat boilers at Garfield, Utah.

The waste-heat boiler plant consists of all units connected to reverberatory furnaces, and receiving heat from the exit gases therefrom. In this classification there are two 490-boiler hp. Erie City vertical water-tube boilers, and six 330-boiler hp. type K-19 Stirling water-tube waste-heat boilers. Each furnace has boilers arranged in battery setting. The boilers are directly in front of the furnace and connection is made through a short uptake and flue (Fig. 1). The Stirling boiler settings rest on concrete foundations at ground level, with the furnace uptake and flue at a low elevation. The Erie boilers are carried on steelwork some 15 ft. above ground level, and stand against the face of the uptake with no connecting flue. The constructional requirements of a larger furnace dictated this type of setting.

* Plant Engineer, The American Smelting and Refining Co.

Owing to the direct path of gas and short flue system, a large amount of dust, slag, etc., is carried over and forms accretions on the tubes, particularly on the front row. This makes it impossible to use baffling of any kind, therefore the boilers act as single pass boilers. Air lances are used almost continually to keep the accretions at a minimum.

In addition to the usual boiler accessories each of these boilers has tracyfiers and traps and two distinct feed-water systems. The main feed-water system supplies treated water from the power house. The secondary system feeds raw water through injectors, and is for emergency use only. Blowoff is taken from the mud drum on the Stirling boilers, whereas surface blowoff is used on the Erie City boilers. Bottom feed with blowoff is found to be most satisfactory. Feed-water regulators have been tried but have not been successful, owing to heavy surges when the furnace is taking charges, boiler-water concentrations and other reasons.

Boiler feed water is made up of condensate returned from turbogenerators, reciprocating engines, power-house auxiliaries and heating steam, plus 25 to 30 per cent of make-up water softened by the Zeolite process. It is heated in open feed-water heaters by exhaust steam to 210° F. and pumped to the boilers, where it is fed at from 180° to 190° F. Table 1 shows the analysis of the feed water at various points in the system.

TABLE 1.—*Analysis of Boiler Feed Water in Grains per Gallon*

Substance	Raw Water	After Softening	Hot Well	After Open Heater	In Boilers
Sodium chloride	30.1	29.8	8.71	8.37	377.0
Sodium sulfate	9.36	9.65	2.04	1.75	117.3
Sodium carbonate	6.78	13.45	3.98	1.46	33.1
Sodium hydrate					82.6
Calcium carbonate	3.9				
Magnesium carbonate	1.8				
Silica	1.8				36.4
Total dissolved solids					629.8
Total alkalinity P	12.5	12.55	3.21	Neutral	118.8
Total alkalinity M					134.4

The concentration of salts in the boiler water as shown in the table is comparatively high. Generally this is kept at from 450 to 500 grains per gallon, although concentrations as high as 650 grains per gallon have been carried without a great deal of priming. The condition of the boiler water is checked at frequent intervals daily by the use of a Leeds & Northrup conductivity meter, and the blowdown is governed thereby. The boilers are blown down, as a rule, at least three times daily to keep the concentration below the limits. If the concentration rises, or for any

reason there is trouble from foaming or priming, more frequent blowdown is made, but this occurs very seldom. When foaming or priming does take place it can be generally attributed to condenser leakage, as the cooling water carries 240 grains per gallon of hardness. Condenser leakage is controlled by daily checks on condensate from all sources. By this practice scale is kept down. It is not entirely eliminated because there are three possible sources of contamination of boiler water by scale-forming salts: condenser leakage, occasional use of injector-fed raw water and operation of the Zeolite softener beyond the time for regeneration.

The boiler performance is checked also by flowmeters. All water leaving the open feed-water heaters is measured by a Lea V-notch meter. In some instances a check is made at the boiler by a Republic flowmeter on the feed water. All steam leaving the boilers, except pop-off, is metered by Republic flowmeters on a 10 and 14-in. header line to the superheater and power house, and a 4-in. heating line. Blowdown is generally from 2.5 to 3 per cent of total feed, and is checked by hot-well and boiler-water concentration. All differences between water fed and steam produced, except above blowdown, is charged to pop-off. This, of course, necessitates periodic checking of all meters.

Steam is generated at 150 lb. per sq. in. gage, or 162½ lb. absolute, and is passed through a separately fired superheater where it is given a superheat of 100° F. This is a Superheater Co. separately fired twin unit, brick setting, with Detrick flat arch covered with Johns Manville superheater blanket, originally built for pulverized coal but at present fired with natural gas. Each section is rated at 75,000 lb. of steam per hour at 110° F. superheat. At present only one unit is operating at from 40 to 60 per cent of rating.

The reverberatory furnaces to which these boilers are connected treat hot roaster calcines and molten converter slag and produce a copper matte varying from 20 to 38 per cent copper. The fuel used on these furnaces is natural gas from the Baxter Basin, Wyoming, and has the characteristics shown in Table 2.

TABLE 2.—*Characteristics of Natural Gas from Baxter Basin Field, Wyoming*

	Per Cent by Volume		B.t.u.
Methane	86.2	Calculated gross heat value at 30 in. and 60° F.	994
Ethane	7.1	Calculated net heat value at 30 in. and 60° F.	893
Carbon dioxide	0.1	Average calorimetric value as purchased at Garfield at 12.6 lb. plus 4 oz. and 60° F.	861
Oxygen	0.1		
Nitrogen	6.5		

From 3000 to 3500 M. cu. ft. of gas is burned per furnace per day through six burners, at a pressure varying from 15 to 20 lb. per sq. in. gage. The gases entering the boiler are at approximately 2300° F., and leaving the boiler are between 800° and 900° F. Table 3 shows the analysis of the flue gases before and behind the boilers. Typical operating results are shown in the heat balance of Table 4.

TABLE 3.—*Analysis of Reverberatory Furnace Gases Before and Behind Boilers*

Item	Before Boilers, Per Cent	After Boilers, Per Cent
Carbon dioxide	10.50	9.04
Carbon monoxide		
Oxygen	1.36	4.64
Sulfur dioxide	1.20	1.04
Nitrogen	86.94	85.28
Excess air	6.50	26.50

TABLE 4.—*Heat Balance of Waste-heat Boiler*

Item	B.t.u. per Cu. Ft.	Per Cent
Calorific value of fuel	876	100
Heat absorbed by boiler	262.5	29.98
Heat loss due to burning H_2 to H_2O	122.5	13.99
Heat loss due to heat in dry flue gas	147.8	16.88
Heat loss due to CO		
Heat loss due to H_2O in air	1.0	0.11
Heat absorbed by furnace	333.8	38.08
Heat loss through radiation and unaccounted for between boiler inlet and outlet	8.4	.96
	876	100.00
Available heat absorbed by boilers, per cent		48.4

Steam pressure, lb. per sq. in. gage	153.6
Barometer, in. of mercury	25.6
Feed-water temperature, deg. F	190
Flue-gas temperature before boiler, deg. F	2,310
Flue-gas temperature after boiler, deg. F	850
Volume of fuel per hour, cu. ft	162,400
Weight of water evaporated per hour, lb	45,650
Gas pressure at burners, lb. per sq. in. gage	15

Inasmuch as the heat absorbed by these boilers from the flue gases is salvage heat, and a byproduct, steam, is produced and used for power production, heat and processing, a certain part of the fuel fed to the furnace can be credited to smelting and charged to power, heat and processing. If the fuel credit were made on the basis of actual heat

recovery, furnace operations would be materially benefited. At the same time power and other users would be penalized for poor boiler operation, something over which they have no control or means of directly improving. Anything tending to improve boiler operation, such as baffling, will hamper furnace operations. In order to reach an equitable basis for this fuel credit, power and other users are charged, on the basis of the steam used, for the amount of fuel that would have been required to produce this same steam in direct-fired boilers.

Application of Refractories to the Copper Industry

By A. G. Suydam,* San Francisco, Calif.

Ancient as is the art of producing copper, so ancient is one of its eternal problems: refractories. Looking backward, in the light of present knowledge, clouded though it be, one cannot avoid a sense of profound respect for the patience, acumen and courage of those who have wrought so successfully despite difficulties, a more accurate understanding of which seems merely to accentuate.

The problem of refractories arises from the necessity for constructing containers in which to bring natural ores into thermal solution in order to effect reduction and separation of the valuable metal content and free it of undesirable contamination while producing a waste slag. The very forces known to produce thermal solution of sufficient liquidity are necessarily destructive of exposed refractory linings in proportions varying inversely with thermal conductivity, directly with the porosity and ease of fluxing of the refractories used, and directly with the temperatures employed. Beyond this, mechanical forces are set up within the refractory lining incident to rapid heating or cooling, shrinkage at high temperatures, and unequal or excessive loading, that frequently pass the elastic limit, causing spalling or distortion. The effects of these forces are often accentuated by the necessity of providing at the same time as hot a bath and as cool a container as are consistent with fuel economy.

Evolution in the Use of Refractories

Clay Brick.—Fireclay and clay refractories had to yield wherever they were in contact with slags rich in basic oxides, first because of the combining power of those oxides with free silica and secondly because of the tendency of alumina to form either aluminates or alumina-silicates of varying complexity. Of no less importance, the incipient fusion point was low as compared with good clay brick of today, while the brick lacked strength at high temperatures. Expansion was slight, therefore it was difficult to make and maintain tight walls, roof, or bottom.

However, the spalling tendency was low because of some of these properties and clay brick have retained their place in roasting furnaces, boiler furnace linings, and in the upper walls of some of the reverberatory types such as some refining, anode and wire-bar furnaces. They have served often and most efficiently for hot patching.

* District Sales Manager, Harbison-Walker Refractories Co.

It is interesting to note early attempts to substitute natural stones, composed principally of the oxides, lime, silica and magnesia.

Silica Brick.—Silica brick, the distinctly acid refractory, has found wide and varying use, primarily because of its strength at high temperatures and its relatively high temperature of incipient fusion, which retards inter-reaction even with basic slags. The development of the ore-melting reverberatory furnace with sprung arches up to 30-ft. span was possible at the time, only because of the strength of silica brick and the careful provision for its expansion characteristics by furnace designers and operators. The pronounced expansion of silica brick at moderate temperatures, i. e., up to about 1200° F., is rather an advantage in securing tight joints, but makes necessary careful heating and cooling through this critical range, to avoid spalling.

The constancy of volume above 1200° F. permits rapid temperature changes above that temperature without damage. Well burned silica brick tend to develop a negligible volume growth due to crystalline inversions and are totally free from shrinkage at all temperatures up to their softening point. These combined properties, together with the fact that an equilibrium is attained in the reaction between silica and cuprous oxide to form cuprous silicate, allow silica to remain in the bottoms of cathode-melting and refining furnaces, despite frequent emptying.

Chrome Brick.—Chrome refractories find a more limited use. Brick made of well selected chromites, in which much of the iron is replaced by alumina and magnesia, approach the neutral state as regards the average slag and are indeed inert to the corrosion of mattes and molten metal. However, the vesicular bond between grains of the brick seems to be dissolved by reverberatory slags or melting-furnace dusts, especially in the presence of reducing atmospheres. Again, despite efficient efforts to reduce their porosity, chrome brick tend to absorb molten matte and metal, the eventual recovery of which from used brick is difficult. On the other hand, chrome brick resist the attack of certain slags better than any other known refractory. Such slags were more frequently encountered in days gone by, and there are those of long experience who would testify to the invaluable service rendered by chrome brick.

Magnesite Refractories possess the advantage of chemical suitability in contact with slags or fused furnace dusts rich in the metallic oxides common to copper metallurgy of today. Their high melting point, high thermal conductivity, and strength at all temperatures below their softening point, together with marked improvements in their manufacture, have allowed them to find increasing use, replacing silica brick wherever the refractory is in contact with liquid slags, matte, oxides, or metallic copper, not in a state of comparative rest. The thermal expansion of magnesite refractories and consequent spalling tendency are

regarded as the main retardants to their use. Early efforts to substitute magnesite brick for clay in the linings of converters were anything but encouraging.[1] In the small converters of the time, magnesite brick spalled badly. Their cost was great, compared to the revenue-yielding siliceous ore-clay mixtures in general use. The courage and determination that made the basic-lined converter possible stand out as notable contributions to the industry.

The average converter necessarily operates during the blow through a temperature range embracing the steepest portion of the magnesite expansion curve. Remarkable betterments of service have attended every positive effort toward reasonable preheating or in maintaining reasonably uniform temperatures during waiting periods. Regardless of these facts, the traditional spalling tendency of magnesite brick had confined their use in other furnaces to those portions of side walls at and below the slag line or metal line where the tempering effect of the bath shielded them from rapid temperature change. Recent tendency is breaking through this traditional barrier. The lining of the holding or anode furnace contains silica, if at all, only in the cap. Kuzell[2] describes the new furnaces at Phelps Dodge Corporation's Calumet and Arizona plant at Douglas, Ariz., where the linings, completely of magnesite, have not needed replacement during the year.

In the ore-melting reverberatory, magnesite brick at first appeared timidly above the slag line in bridge walls, in side walls of the hot end, in uptakes, and around charge holes. In the last few years they form roof sections 30 to 50 ft. long and 4 to 5 ft. wide surrounding the charge-hole areas, replacing silica brick because of their markedly superior resistance to the fluxing action of furnace dusts. Likewise, in the refining furnace and in the cathode-melting furnace, magnesite has replaced silica brick in the walls above the metal line to the skews, completely in a number of plants and in others except opposite charging doors. The avidity with which cuprous oxide[3] dusts unite with silica to form cuprous silicates, as compared to the inertness of magnesite, prompts some operators seriously to consider magnesite brick for at least a portion of the crown.

Metalkase Brick.—Twenty years ago it was recognized that one might avail himself of the tensile strength of steel to minimize the effects of spalling. Dead-burned magnesite, provided with a chemical bond, was pressed into steel jackets of either circular or rectangular cross-section. When exposed to high furnace temperatures, the steel burns and melts

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[1] H. A. Keller: *Trans.* A.I.M.E. (1913) **46**, 474.

[2] C. R. Kuzell: Part Time Operation Brings Problems to Smelters. *Min. & Met.* (1933) **14**, 45–48.

[3] O. H. Hoffman and C. R. Hayward: Metallurgy of Copper, 56–57. New York, 1924. McGraw-Hill Publishing Co.

at the surface and is absorbed into the pores of the refractory, making a monolithic surface, while enough heat is lost by radiation to preserve the strength a few inches back from the exposed face and prevent rupture due to thermal shock. These brick have provided a basic lining of low spalling tendency and have rendered a most valuable service.

In the copper industry, Metalkase brick have served best along the converter tuyere line. It is necessary to make the brick according to tuyere spacing. The space between tuyere pipes and the adjacent Metalkase is rammed solid with Thermolith, an air-setting, neutral

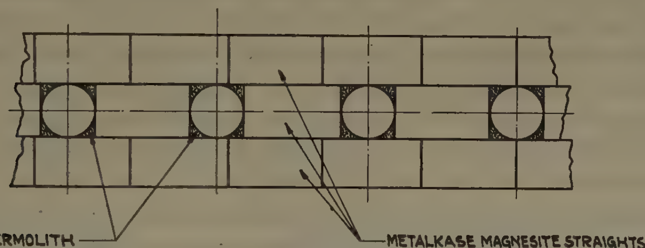

FIG. 1.—METHOD OF USING METALKASE MAGNESITE BRICK AROUND TUYERES.

high-temperature cement. When adequate allowance is made for expansion of the surrounding magnesite, a tight, nonspalling tuyere line is provided. Records of long tuyere-section life have been afforded by this construction, illustrated in Fig. 1.

Partial advantage of Metalkase construction has been secured by the use of steel strips placed between magnesite brick. Best results are obtained by using thin sheets (14 to 16 gage) along both horizontal and vertical joints of walls or along both longitudinal and lateral joints of roofs. The fusion of the steel at the surface causes some adhesion, while the remainder is protected from fusion by radiation and from oxidation by the coverage of the magnesite refractory.

Magnesite the Last Step in Evolution

Summarizing the evolution in the choice of refractories, one finds a general replacement of clay brick by silica or siliceous mixtures in the majority of smelting furnaces, with a later replacement of silica brick by magnesite wherever the refractory is in contact with molten slag, matte, or metal not in a state of comparative rest, and lately in areas of severe dust attack.

The primary influences prompting these changes are the combining power of basic oxides with available silica, and the ever-increasing temperatures employed. Back of these primary causes is the contributory influence of ore dressing. In its evolution from hand-sorting, jigging, table concentration and flotation, it has presented a smelter charge in

successively finer states of division and higher degrees of concentration. The effect was checked by sintering so long as available sulfur remained.

The operator of the average smelter is keenly conscious of a decreasing ratio of sulfur to copper in his charge. The roaster, when used at all, serves as a drier, or, at most, barely to "crack" the charge. As a consequence, an increased proportion of the total work is being placed on the reverberatory furnace. Since it appears that heat is transmitted to the charge in a reverberatory furnace primarily by radiation, and since the rate of transfer varies with the fourth power of the absolute temperature, the tendency is to push temperatures, especially in the ore-melting furnaces, as high as is economically possible.

With the combination successively of finer grind, more basic charge, and higher temperatures, the critical temperature-slag limit of silica brick is soon passed in the smelting end of the reverberatory. Oldright and Schroeder[4] note that the maximum practical temperature at which the furnace may be operated is determined by the point at which the silicates made by dust in contact with silica brick become easily fusible. This temperature was determined for one furnace at 2600° F. It becomes apparent, then, that a refractory better suited to the particular service than silica brick is required for the roofs over the hot end of an ore-melting reverberatory, especially.

Trials of high-melting-point refractories indicate none of more promise than magnesite. In a test conducted at the El Paso plant of the American Smelting & Refining Co. at and above the slag line, high-alumina brick were the first to fail when tested in panels along with chrome brick, first-quality fireclay brick, magnesite brick and silica brick. Petrographic examination[5] of the fused surface disclosed the reason to be the formation of complex aluminum silicates with the constituents of the furnace dust. Silicon carbide brick, while a splendid refractory in many services, failed in a similar panel test at Tacoma, owing to the fluxing action of basic oxides of the furnace dust. Brick made in Harbison-Walker Refractories Company's laboratories, containing high percentages of mullite, while showing remarkable strength at high temperatures, failed to resist the fluxing action as well as did first-quality fireclay brick.

Magnesite in Suspended Roof Construction

Turning now to the trend of development of reverberatory furnaces, one notes a tendency toward greater widths, with decreasing lengths. While this tendency may be checked by the effect of higher ratios of concentration, leaving less tonnage to be smelted, still, the economies

[4] G. L. Oldright and F. W. Schroeder: Suggested Improvements for Smelting Copper in Reverberatory Furnace. *Trans.* A.I.M.E. (1928) **76**, 461.

[5] R. D. McLellan, Perth Amboy, N. J.: Private communication.

incident to dividing larger tonnages into rather constant furnace losses remain in any plant operating more than one furnace. This fact, coupled with recent interest in suspended roofs, focused inquiry upon the possibility of using magnesite brick in this type of construction.

Almost nothing was known regarding the behavior of magnesite brick in suspended construction; the time required for safe heating and cooling; limiting temperatures at which hot patching could be attempted; the effects of relatively high thermal conductivity on hangers; probable heat losses; or the possibilities of insulation. In order to secure needed data, an experiment was conducted at the plant of Harbison-Walker Refractories Co. at Hays, Pa. Early in 1931, an experimental furnace, Fig. 2, was provided with a flat, suspended magnesite brick arch, made

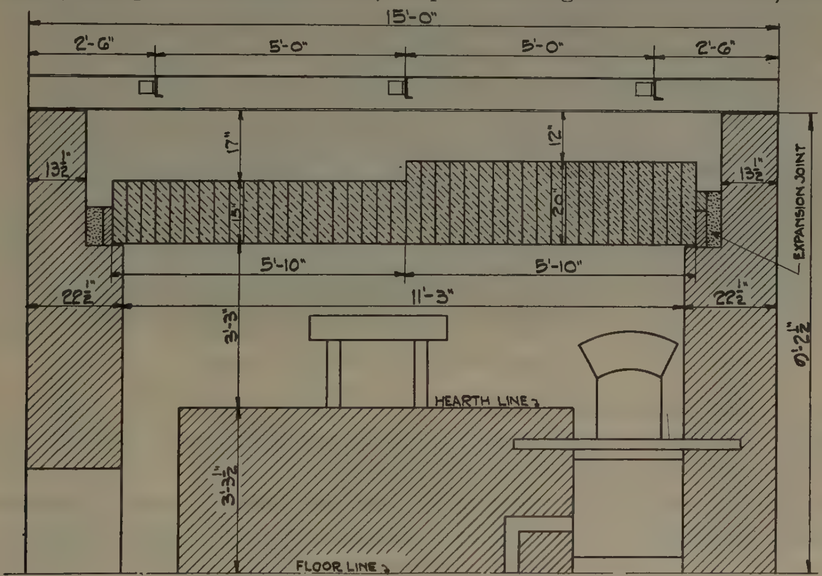

FIG. 2.—MAGNESITE SUSPENDED FLAT ARCH.

in two sections, one of 20-in. and the other of 15-in. length brick. Each section was 5 ft. 10 in. long by 5 ft. $7\frac{1}{2}$ in. wide, and contained 30 hangers of 10 bricks each, making a total of 600 pieces.

Temperatures were taken at 12 points, divided into three groups of four (all approximately on the center line of the furnace). Two groups were located in the 20-in. section and one in the 15-in. section. Each single group consisted of four platinum, platinum-rhodium couples, one projecting through the brick 2 in. into the furnace, the others embedded in the brick and extending to within 1 in., 3 in. and 5 in. of the under or exposed face of the brick. The central group of thermocouples was connected to a recording pyrometer. Temperatures at other points were determined at hourly intervals. The central thermocouple projecting into the furnace was used for control.

The furnace was fired with natural gas, of 1000 B.t.u. per cubic foot, fed through one No. 2 and two No. 7 Maxon-Premix blowers. Each No. 7 blower was fitted with two 5-in. burners. The No. 2 blower was fitted with two 2-in. burners, installed immediately under the two central 5-in. burners. The No. 2 blower was used in initial heating to temperatures slightly above the minimum range of one No. 7 blower.

The test was divided into four runs. During each of these the furnace temperature was raised, held somewhat beyond the time necessary to reach equilibrium, and lowered. The time of heating and cooling was shortened successively in order to subject the roof to thermal shock of increasing severity, until some damage should result. The measure of these cycles and time necessary to reach equilibrium after maximum temperature was attained is shown in Table 1. After completion of each run, the furnace was entered, each roof brick carefully examined, and photographs taken.

TABLE 1.—*Measure of Cycles and Time for Reaching Equilibrium*

Run No.	Temperature Inside Furnace Raised			Equilibrium Reached in	Held at Maximum for	Temperature Lowered		Spalling
	From	To	During			To	During	
1	Atmospheric	2700° F.	58 hr.	16 hr.	100 hr.	Atmospheric	58 hr.	None
2	Atmospheric	2700° F.	25 hr.	13 hr.	30 hr.	Atmospheric	25 hr.	None
3	Atmospheric	2700° F.	12½ hr.	16 hr.	90 hr.	Atmospheric	12½ hr.	None
4A	Atmospheric	2700 °F.	10 hr.		16 hr.	2000° F.	3 hr.	None
4B	2000° F.	2850° F.	5 hr.	13 hr.	30 hr.	No drop		1½ in. at 2850° F.
4C	2850° F.	3000° F.	3 hr.		8 hr. 3 hr.	2800° F. After patching	Patch	1½ in. at 3000° F.

Spalling.—There was no spalling or cracking observable upon completion of the third run. Fig. 3 reproduces photograph then taken. The brick had been heated to 2700° F., cooled to room temperature three times, and exposed to 2700° F. for a total of 220 hours. During run 4A no spalling was observable from furnace doors. When spalling began, during runs 4B and 4C as temperature approached 2850° F., it was apparent from furnace doors. It is worthy of note that no spalling whatever appeared during the first three runs, or during the fourth run until after a temperature of 2850° F. had been reached. It appears probable that this spalling, in a measure, was due to the development of horizontal planes of weakness caused by shrinkage of the hot ends of the brick, by lateral pressure or by both. The fact that not any spalling occurred during run 4A, in which the roof was raised from cold to 2700° F. in 10 hr.,

or at the rate of about 260° F. per hour, should be interpreted with caution, since this, based upon experience, is considered a rapid rate of heating for magnesite brick.

Insertion of Cold Shapes in Hot Roof.—During run 4A and 10 hr. after 2700° F. had been reached, gas and air were shut off for 4 min. while two

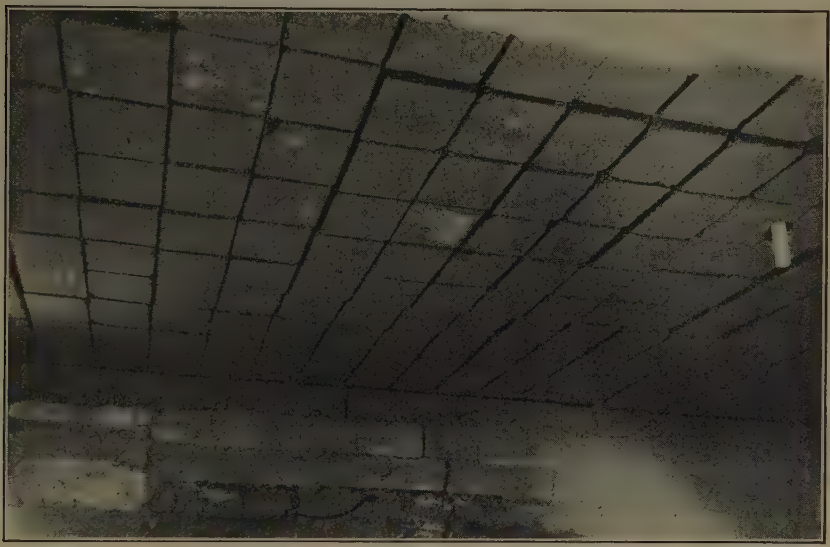

FIG. 3.—TEST ROOF AFTER THIRD RUN.

magnesite shapes were withdrawn from the roof and replaced by new cold shapes. The old brick were sound and no spalling of the new shapes was observed. Again, at the end of this run, when furnace temperature had been lowered to 2000° F., a second pair of roof shapes was replaced by new cold shapes. Again no spalling was evident. It is felt these tests also must be interpreted with caution. Owing to the position of the superstructure of the test arch, no larger patch could be attempted.

Furnace-door Spalling Test.—During the heating period of run 4A, a spalling test was conducted by placing in the door of the furnace six sets of three each of the 20-in. magnesite roof shapes, at temperature intervals of 300° F. On the heating schedule, 1 hr. 15 min. was required for each interval, hence the set put into the furnace at 2700° F. was also heated for that period. After heating, each set of shapes was removed and allowed to cool in the air, and the next set inserted. The brick put in at 1200° F. and raised to 1500°, in at 1500° F. and raised to 1800°, and in at 1800° F. and raised to 2100° showed no spalling or cracking. Those heated from 2100° to 2400° F. all showed end cracks, and one lost a small corner. The set heated from 2400° to 2700° F. developed end cracks and two of them spalled. The last set, heated at 2700° F. for 1¼ hr., all spalled and developed end cracks. This cracking occurred when brick

were placed in the furnace. Apparently rapid cooling in air had no noticeable effect. The total loss of weight by spalling of the last three sets was 0.2 per cent, 1.2 per cent and 3.2 per cent, respectively.

Thermal Expansion.—The over-all expansion after reaching equilibrium at 2700° F. was 0.78 in. for the top of the arch and 1.00 in. for the

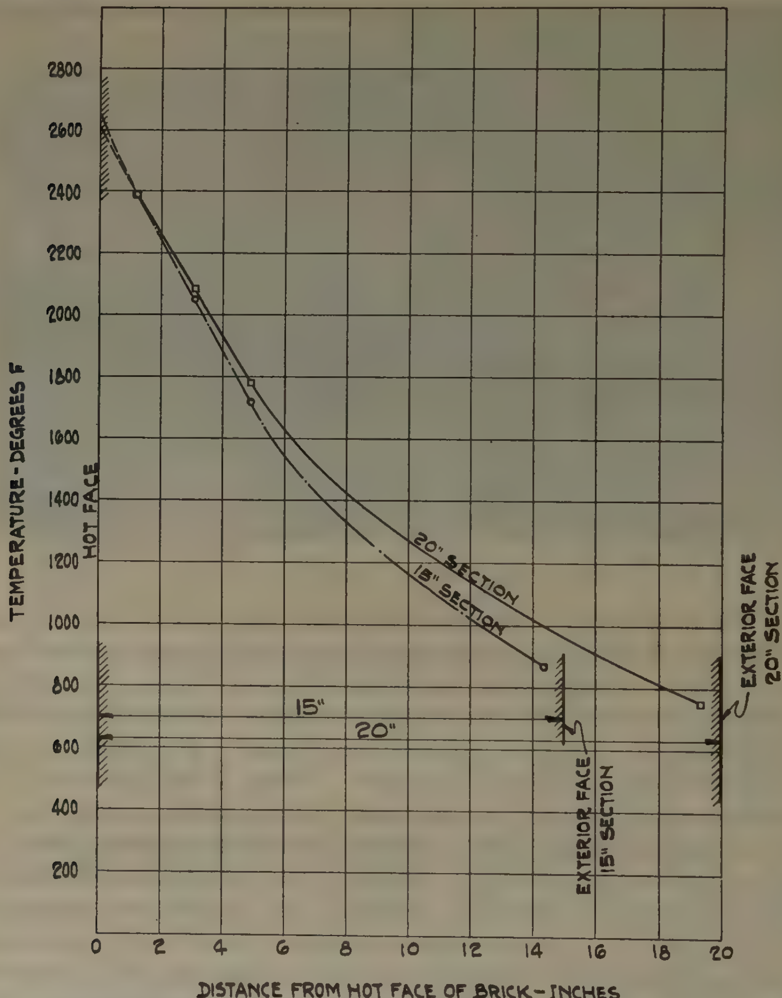

FIG. 4.—TEMPERATURE GRADIENT THROUGH BRICK AT EQUILIBRIUM AT FURNACE TEMPERATURE OF 2700° F. (AVERAGE OF 25 HOURS AT EQUILIBRIUM).

bottom, this corresponding to 0.069 in. per foot at the top and 0.089 in. per foot at the bottom. Since calculated expansion for a solid block arch of the same size would be considerably more, it may be assumed that joints, 40 in all, each took up approximately 1/26 in. of the actual expansion.

Shrinkage.—At the end of the third run, four hangers holding 40 bricks were removed from the center row of the 15-in. section, being the

second, sixth, ninth and tenth from the front of the section. All bricks were carefully examined and found to be free from cracks or spalls. These bricks had been heated to 2700° F. and cooled three times, and had been held at 2700° F. for a total of 220 hr. They were carefully measured for shrinkage. The average measurements are: length, 14.97 in.; width at top, 4.52 in.; width at bottom, 4.44 in.; thickness at top, 3.56 in.; thickness at bottom, 3.47 in.

Comparison with Silica Brick.—Prior to the fourth run, two hangers, each holding ten bricks, were removed from a central position at the stack end of the 15-in. section, and replaced by silica bricks of identical shape. A parting joint of Thermolith, a chrome-base air-setting high-temperature cement, was used between the silica and magnesite shapes. This section remained in place during runs 4A, 4B and 4C.

Temperature Gradient.—In Fig. 4 are plotted the measured temperatures through each thickness of roof shapes. Heat losses through the roof and side walls of a reverberatory furnace, constructed of whatever refractory, are obviously small compared with losses through the stack and the setting and in the slag, matte, or metal tapped. This seems substantiated by the report of one operator, that upon substituting magnesite for silica in approximately 30 per cent of the roof over the hot end of an ore-melting reverberatory furnace, no change in fuel ratio was apparent.

Magnesite Brick in Sprung Arches

If one becomes convinced of the chemical suitability of magnesite and concludes he can operate his furnace so as to avoid spalling, he may yet hesitate to substitute magnesite for silica throughout sprung arches of large span because of the increase in weight of the arch. Calculations of skewback pressures, even for cold arches, cannot be accurate, since they depend upon two fundamental hypotheses, which may be considerably in error.[6] However, following through the calculations, one observes that however indeterminate the stress distribution may be, the compressive stress at the skew remains a vector quantity depending upon the weight of the roof. For an arch of given dimension, this relation must be linear. Consequently, the compressive stresses in arches of silica and magnesite would be in the ratio of their respective weights, approximately 6 to 10. Combinations of silica and magnesite, such as have been used, would involve stresses in proportion to relative combined weights. Such calculations may assist the furnace operator to judge whether increased loading, due to substitution of magnesite brick, would exceed the safety factor provided in his furnace design. Thus, calculation of cold arch stresses figured for a sprung arch 18 in. thick over a 23-ft. 9-in. span with a 2-ft. $5\frac{11}{16}$-in. rise gave skewback pressure as follows:

[6] W. Trinks: Industrial Furnace, **1**, 194–198. New York. John Wiley & Sons.

(A) Arch of all silica brick.. 21.4 lb. per sq. in.
(B) Arch of magnesite brick 4 ft. out from each skew, central portion of silica.. 25.9 lb. per sq. in.
(C) Arch of all magnesite brick.. 35.7 lb. per sq. in.

During operation with roof B, buckstay pressures were raised to a point that, in the opinion of the operators, undoubtedly would have carried an all magnesite roof without any sign of crushing.

At another plant, where magnesite brick of not a hard burn was used in a similar construction, the magnesite section spalled with measurable shrinkage until only 4 in. of thickness remained. Yet this thin section supported the silica section which, at the center, was all of 10 in. thick. The pressure per unit of area at the skews was considerably more than that resulting from an all-magnesite sprung arch of uniform thickness. Evidences of this sort indicate that magnesite brick will carry their own weight over spans at present used, provided one face of the refractory is cooled by radiation to air.

In this connection, relative crushing strengths are of interest. Unfortunately, such tests as have been made have subjected the refractory to soaking heats, a condition met in practice usually in the bottoms only. Nesbitt and Bell[7] give the average of 10 tests each on clay and silica brick as shown in Table 2. The data in Table 3 were compiled from our own records of standard A.S.T.M. load tests made under soaking heats and loads of 25 lb. per square inch.

TABLE 2.—*Crushing Strengths of Clay and Silica Brick* (*Nesbitt and Bell*)

Refractory	Crushing Strength, Lb. per Sq. In.			Load Required to Cause First Deformation, Lb. per Sq. In.	
	30° C. 86° F.	1000° C. 1832° F.	1350° C. 2462° F.	1000° C. 1832° F.	1350° C. 2462° F.
Clay A	1668	2892	1289	1458	576
Clay B	1743	4258	948	No deformation before fracture	669
Silica C	3965	2148	1831	No deformation before fracture	
Silica D	1700	1587	850	No deformation before fracture	

Since neither hard-burned magnesite nor silica brick show deformation before failure, but the cold strength of magnesite averages considerably above that of silica and the steep temperature gradient through the average roof allows such a large portion of the arch thickness to retain its "cold" strength, it seems reasonable that magnesite should possess adequate strength to carry its own weight. The real concern then centers

[7] *Proc.* Amer. Soc. Test. Met. (1918).

in the consideration of over-all expansion and whether an all-magnesite roof over large spans may be brought up to temperature without buckling. This is a subject upon which one would wish to proceed with caution, being sure to provide a sufficient thickness in relation to span. At one plant where a 9-in. magnesite roof of comparatively moderate span buckled, a 13½-in. roof subsequently installed gave no trouble whatever.

TABLE 3.—*Crushing Strengths of Various Types of Brick*

Refractory	Cold Crushing Strength, Lb. per Sq. In.	Deformation under Load of 25 Lb. per Sq. In., Per Cent		Temperature at Which Brick Fail under 25 Lb. Loading[a]
		1350° C. 2462° F.	1450° C. 2642° F.	
First-quality clay brick (medium to hard burn)	900–2000	4–7	10–20	
First-quality clay brick (hard burn)	1200–2500	2–6	7–20	
50 per cent alumina (diaspore)	1500–2500	1–2	6–8	
60 per cent alumina (diaspore)	2100–2800	0–1	4–6	
70 per cent alumina (diaspore)	4000–5500	0–0.7	3–4	
60 per cent alumina (cryst. Al_2O_3)	5000–7000	Negligible	1.5–3	
Silica brick	1200–2500	[b]	[b]	2900° F.
Magnesite (regular)	2500–6500	[b]	[b]	2500° F. to 2600° F.

[a] Heating schedule standard A.S.T.M. load test to 1350° C., then 100° C. per hour until failure.

[b] Silica and magnesite brick usually fail abruptly without previous deformation.

That the subject is being approached with caution is indicated by the gradual replacement of silica by magnesite in a number of reverberatory roofs. From a beginning, consisting of a trial of a few brick immediately surrounding the charge holes, experimentation has widened the replacement until magnesite sections 20 courses out from each skew and covering the length of the smelting zone are now being tried. So far, fluxing of the central section, built of silica brick, has terminated the roof life, however far from the charging zone that section has been removed.

VARIATIONS IN TYPES OF BRICK

Good unburned magnesite brick are being made with a chemical bond. The ground mixture with dead-burned magnesite is sized, pressed and dried. Obviously it is impossible to force together grains of a material like magnesite so closely that further union may not occur on sintering. Whether heat be from intentional firing in the kiln or from exposure to service conditions, shrinkage on firing occurs. If this shrinkage be non-uniform, as is unavoidable except under kiln-firing conditions, strains are set up, which cause what is known as "shrinkage spalling." This strain occurs in any refractory having an appreciable expansion characteristic

when subject to nonuniform shrinkage at high temperatures. It is the primary cause of spalling more often than is commonly realized.

Unfortunately, there is available no ready method of measuring the precise effect of furnace temperatures on this type of brick because of the temperature gradients almost always present in practice. It may be assumed, however, that the chemical bond is gradually replaced by a sintered bond, extending from the hot face outward a distance inversely proportional to the slope of the temperature gradient.

Fig. 5 shows the effects of reheating on the modulus of rupture of typical chemically bonded brick. Hard-burned magnesite brick do not

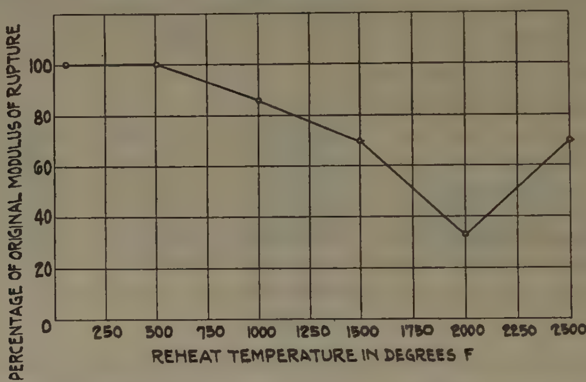

FIG. 5.—EFFECT OF REHEATING ON MODULUS OF RUPTURE OF TYPICAL UNBURNED MAGNESITE BRICK AT ATMOSPHERIC TEMPERATURE.

lose their strength when reheated at these temperatures but rather tend to become stronger. The curve suggests a progressive weakening from the beginning of breakdown of the chemical bond until sintering begins. If the refractory may be so used as to guard against the adverse effects of this yet unavoidable property and such shrinkage as is corollary, good service may be secured. The advantages of the refractory are found in lower first cost and a lower spalling tendency when subjected to thermal shock before sintering begins. The utility of the unburned magnesite brick often is enhanced by the use of steel sheets in the joints, the function of the steel being the same as in the Metalkase magnesite brick. The additional cost of the steel is well warranted in many applications.

Low-iron Magnesite.—For the past 16 or 18 years, a magnesite brick, the iron content of which is appreciably below the 4 to 5 per cent Fe_2O_3 of regular magnesite brick, has been manufactured on a limited scale and for a limited use. Not until the causes of deterioration of regular magnesite brick were more clearly understood did the probability that this refractory might render a better service anywhere in the copper industry become suspected. Not until manufacturing difficulties were markedly reduced was the cost of the refractory brought within commercial limits. Laboratory tests and some actual service records would indicate this

newer type of brick to possess some decided advantages in a number of the various applications to the copper industry.

Power-pressed Silica Brick.—Efforts to improve silica brick from a chemical standpoint have yielded little. The migration of lime described by Oldright and Schroeder[8] had been noted in the open-hearth roof, where the added effect of copper oxides was not involved. Whether that migration is due to the lower specific gravity of the calcium glass as compared to ferrous silicate; to a greater liquidity under given temperature-pressure conditions, permitting further absorption by capillarity, or to immiscibility between the two glasses, is not known.

The system $FeO.SiO_2$ has been studied.[9] Much work has been done on the various lime-alumina-silica mixtures, with or without iron.[10]

Silica brick when cold normally contain about 10 per cent of glass. At high temperatures, the amount of liquid phase will vary with the percentage of iron present; up to 40 per cent FeO lowers the melting point of $FeO.SiO_2$ mixtures only 24° C. but the amount of glass increases rapidly. (Additional silica is carried into the glass phase by absorption of lime and alumina from furnace dusts.) So long as the brick may receive the bases from the furnace dusts, glasses will be formed from their fusion and will penetrate the brick so far as the temperature gradient and porosity of the brick will permit.

The porosity of normal silica brick averages about 25 to 28 per cent and even higher. A method has been found which allows the heavy power press to be used in the forming of a considerable variety of silica shapes. As a result, the porosity of the finished product is lowered appreciably. A more pronounced gain was made in workmanship. As a result of power pressing and improved methods of firing, dimensions of the finished brick are considerably more uniform. Warped surfaces virtually have been eliminated. As a result, considerably better life is being received from silica roofs wherever the severity of service is not subject to constant increase.

Other Refractories

Acidproof Brick.—Acidproof brick have been used for years in the linings and packing of acid towers, the linings of acid-cooling and storage tanks, and as insulators between electrolytic tanks and their supports. More recently, some thought has been given to the reduction of electric current leaks above the tanks used for electrolytic refining of copper.

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[8] Reference of footnote 4.

[9] Bowen and Schairer: *Amer. Jnl. Sci.* (Sept., 1932). C. H. Herty, Jr. and G. R. Fitterer: *Ind. & Eng. Chem.* (1929) **2**, 51–57.

[10] For a rather complete reference, see F. P. Hall and Herbert Insley: A Compilation of Phase-Rule Diagrams of Interest to the Ceramist and Silicate Technologist. *Jnl. Amer. Cer. Soc.* (1933) **16**, 538–540.

Capping boards commonly used on tank walls become saturated with electrolyte, the conductivity of which is high.

Duro acidproof 9 by 4½ by 1⅛-in. brick were tested by being soaked for three days in electrolyte, washed and dried. They were then placed between two copper sheets attached to Megger terminals, and loaded to 20 lb. to secure contact. Average resistance at full Megger pressure of approximately 500 volts was about 100,000 ohms. The brick were then supported at each end. A piece of 1-in. square iron was placed on top of each brick at the center and loaded with 1000 lb. of pig lead. The brick did not break under this load and were judged suitable for replacement of capping boards, when made to exact dimensions required.

High-temperature Cements.—The development of better materials for bonding brickwork has added considerably to furnace life. Cements may be selected so as to prevent inter-reaction with the brick they join and are not dependent upon high temperatures for their initial set. Not a little has been added to the life of converter brick by the use of such a cement, especially at and near the tuyere line, where effects of abrasion are severe.

In construction of roaster hearths it has been possible practically to exclude the infiltration of calcine into the brick joints by causing a good strong cement to bridge the joint. Air leakage is considerably reduced by the judicious use of suitable cements.

Looking Forward

The copper industry will not halt its constant march toward higher efficiency. In looking forward to the changes that march will prompt, one engaged in the refractory business cannot avoid the consciousness of the need ever to better his products, to meet the demands of ever-increasing severity of service on the one hand and the insistent demands of economy on the other. Unfortunately, these commands are voiced in opposite tongue and compromises too often may be necessary.

The limits of known available raw materials seem almost reached. Insight into the causes of failure is clouded primarily by a lack of certain knowledge of the complex chemistry of thermal solution and secondarily by the fact that one may critically examine the effect only in its cold state. Too often crystallizations and separations on cooling disguise the true story and return a false answer. However, the laborious and patient work of those devoted to research along lines involving thermal solution has contributed much to the understanding of fundamentals and may shed much light on many phases of the subject about which little is known now.

The patience and forebearance of those who use refractories and their cooperation in testing and diagnosing the causes of the many destructive forces is of the utmost help.

Summary

1. The trend in the choice of refractories has been toward the more basic types as smelter feed has become more highly concentrated.

2. Fine grinding incident to flotation, and higher temperatures necessary to lower costs have imposed successively harder conditions, especially in the ore-melting reverberatory.

3. The tendency to increase furnace widths and temperature of firing makes almost imperative the selection of a refractory better suited to the service than is the commonly used silica brick.

4. Of known refractories, magnesite brick seem to possess the most desirable properties. Their high expansion characteristic, weight and cost have prompted careful increases in the proportion of magnesite substituted for silica in sprung arches of large span.

5. Nothing was known of the behavior of magnesite brick in suspended construction. Such a roof was provided over an experimental furnace and data gathered regarding the following points: period of safe heating and cooling; temperatures at which cold shapes could safely be installed; temperature gradient through brick; probable heat losses through the roof; over-all expansion; the effect of commonly used expansion joints and some information on the probable causes of spalling.

6. The probability of magnesite brick serving as a complete sprung arch over wide spans is considered in the light of such data as are available. Ideas as to the strength of magnesite brick at high temperatures may have been clouded by the results of tests under soaking heat conditions without due consideration of the mechanical strength of the refractory as actually used in practice, where a rather steep temperature gradient is had except in bottoms.

7. New types of magnesite brick have been evolved. One is a hard-burned low-iron brick now in the process of commercial testing. The other is a chemically bonded brick, the chief advantage of which is a lower cost.

8. The insulating value of acidproof brick is being utilized in the refinery.

9. High-temperature cements have helped in overcoming the difficulties experienced formerly with clays.

10. Silica brick have been improved by adapting to them the power-press method of manufacture. The gain is principally in workmanship.

Refractories

By R. P. Heuer,* Philadelphia, Pa.

In recent years an increasing amount of research work has been done on refractory materials for use in copper-smelting furnaces. A few of the larger refractories manufacturers have erected special research laboratories which are fully equipped to study the refractory problems of the smelter. A typical laboratory of this type was described in the January, 1929, issue of Mining and Metallurgy, in an article entitled "Methods of Research Newly Applied to Refractories," by W. F. Boericke. This particular laboratory is equipped for the manufacture of full-size refractory brick. It aims to study on a fairly large scale the behavior of various refractory raw materials and the effect of improved methods of manufacture.

The refractories manufacturer has at his disposal the naturally occurring refractory oxides in various degrees of purity, mixed or combined with other oxides. In order to supply the copper smelter with a refractory brick that will permit the most economical operation of the smelting furnaces, it is necessary to have the optimum combination of a number of both physical and chemical properties in the brick. Unfortunately, nature has not provided any raw materials in which the perfect combination of these properties exists. It is, therefore, a problem for man's ingenuity to seek out the most desirable raw materials and develop methods of manufacture that give, in the resultant product, the necessary combination of properties in as full a measure as is possible.

In the laboratory it is possible to carry out fairly broad experiments with this objective. It is first necessary to obtain and process the raw materials, then to manufacture them into brick and finally, to test these bricks under conditions designed to interpret or simulate in the laboratory the actual conditions in the smelting furnace. It is necessary for the research engineers to determine certain fundamental properties in the brick which may be subjected to laboratory measurements and to interpret these measurements in terms of brick performance in smelting-plant practice. Such procedure is not what might be called an exact, scientific one but it has been found that great progress can be made when these principles are applied. For example, the resistance of refractory brick to spalling and thermal change can be related to the thermal expansion of the refractory, its thermal conductivity and heat capacity and its

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resilience as measured by the ratio of modulus of elasticity to modulus of rupture. The resistance of refractory to slag attack can be gaged to some extent by the melting point of the refractory, its porosity, its chemical and physical behavior with slags with which it comes in contact. Its chemical and physical behavior can be further studied by petrographic examination of brick samples that have been exposed to such slags.

Recent developments in the manufacture and application of new magnesite refractories to copper-smelting furnaces are a typical illustration of the results obtainable in this type of research. In the December, 1931, issue of MINING AND METALLURGY A. E. Fitzgerald described "New Developments in Unburned Magnesite Brick for the Metallurgical Industry." In this article a new principle for the manufacture of magnesite brick was described. Bricks so produced were found to be much cheaper to manufacture and at the same time had improved resistance to spalling and the other destructive forces that prevail in many metallurgical furnaces. The developers of the brick felt that with the new brick they could accomplish two purposes: (1) because it was a cheaper brick, it could be substituted for the more expensive magnesite refractory, which theretofore had been manufactured; (2) because it had more desirable properties than the older type of magnesite brick, it could be used in places where the older brick had failed because of certain deficiencies in its physical characteristics.

Practical developments since the appearance of the new brick in 1931 have shown that it could be substituted for the older and more expensive brick in almost all places. The new brick could also be used in many places where the older brick failed. As an illustration of new uses for the unburned magnesite brick may be cited the construction of a magnesite suspended roof for the smelting reverberatory of the Hudson Bay Mining & Smelting Co., Flin Flon, Manitoba, Canada. Before the installation of the unburned magnesite brick the average life of a silica-brick roof in the hottest section was 54 days. The frequent repairs, necessitating loss of production, extra expense for coal and labor for firing waste-heat boilers, etc., were both inconvenient and costly. The installation of a roof that would allow higher operating temperatures, last longer and provide for a larger daily melting capacity seemed very desirable. The silica brick was unable to stand the corrosive action of the dusts with which they came in contact in this furnace roof. It was recognized that the magnesite roof would be more refractory, and more resistant to the corrosion of these dusts than silica.

The important thing in the magnesite roof was to get a brick and a type of roof construction that would not spall or fail under the mechanical forces set up in such a large roof span. A mechanically suspended roof for a section 27 ft. long in the hottest zone of the furnace and extending the full width of the furnace (21½ ft.) was constructed. On the first

campaign this roof section ran 259 days. It became necessary to shut down the furnace, not because of the failure in the unburned magnesite section but because the silica in a cooler section of the roof had been worn

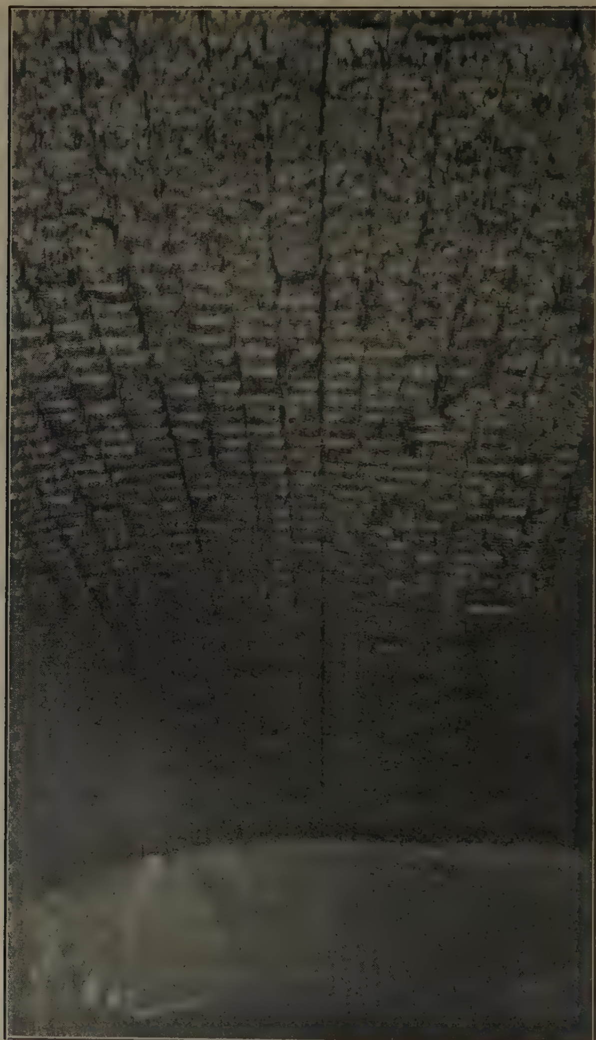

Fig. 1.—Condition of magnesite suspended roof in reverberatory of Hudson Bay Mining & Smelting Co. at Flin Flon, Manitoba, after repair.

through. This silica section was replaced by more unburned magnesite brick and the total area of unburned magnesite in the roof then became 21½ by 51 ft. Fig. 1 shows the condition of the roof after the repair. In the foreground is the unburned magnesite that has been through the 259-day campaign. In the background can be seen the newer installation

of unburned bricks that have replaced the silica bricks that failed. After this repair the furnace was again started and at the present writing is more than one year old and still in operation. It is interesting to note that this is the first successful installation of a suspended magnesite roof to a large copper-smelting reverberatory furnace. This illustration is cited to typify the results obtainable by cooperative research between refractories manufacturers, furnace engineers and copper-smelter operators.

Repairing a Large Smelter Chimney Injured by Spalling

By John E. Lanning,* Clarkdale, Arizona

During the last decade, since most smelters have included Cottrell plants as standard equipment for the removal of dust from furnace gases, it has become apparent that smeltermen have had a new problem slowly developing. This problem has manifested itself in accelerated destruction of, or serious injury to, the masonry of flues and chimneys. The masonry either spalls, if the material is of low strength, or acquires large major cracks, either of which conditions attains such proportions that renewals or reinforcing become necessary to maintain the structure in satisfactory condition.

This paper describes the failure of a large modern brick chimney to such an extent that its safety was vitally impaired; the investigation of its condition; the methods of restoring satisfactory strength to the structure; and the installation of an acidproof lining.

Plant History

The Clarkdale smelter of the United Verde Copper Co., which was put in operation in 1915, was primarily a blast-furnace plant. The smelter at Jerome had utilized blast furnaces exclusively, and in the design of the new (1915) plant, blast-furnace smelting evidently was considered of major importance, the smelting equipment consisting of four blast furnaces 48 in. by 26 ft. 8 in. and three reverberatories, each 19 by 101 ft. The latter, supplemented by roasters, were intended mainly for handling dust and fine ore resulting from the crushing operations. The original roasting equipment consisted of six six-hearth Wedge roasters, six more being added in 1917. All gases from blast furnaces, reverberatories, roasters and converters were conducted by means of suitable flues and dust chambers to one central chimney.

This chimney, 30 ft. inside diameter by 400 ft. high, is a self-supporting steel structure, lined with 4-in. brick supported on steel shelf angles, riveted to the shell, spaced about 15 ft. vertically. The bricks were of local manufacture, not particularly hard-burned, and were laid in ordinary cement-fireclay-sand mortar. No record is available to indicate that any particular effort was made to secure an acidproof mortar. The same materials were employed in the construction of the reverberatory and roaster flues, and the roaster dust chamber. Parenthetically, it may

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be stated here that the chimney lining is today in almost perfect condition. This will be referred to later in this article.

About 1918 it became evident that dust losses were excessive, and with the trend increasingly toward reverberatory smelting, a thorough investigation of dust and fume conditions was undertaken. As a result, a Cottrell precipitating plant and chimney were built coincident with the construction of three new 25 by 101-ft. reverberatories and the installation of 12 new seven-hearth Wedge roasters. Two of the new reverbs were blown in in 1920, and the 12 roasters and the Cottrell treater and chimney

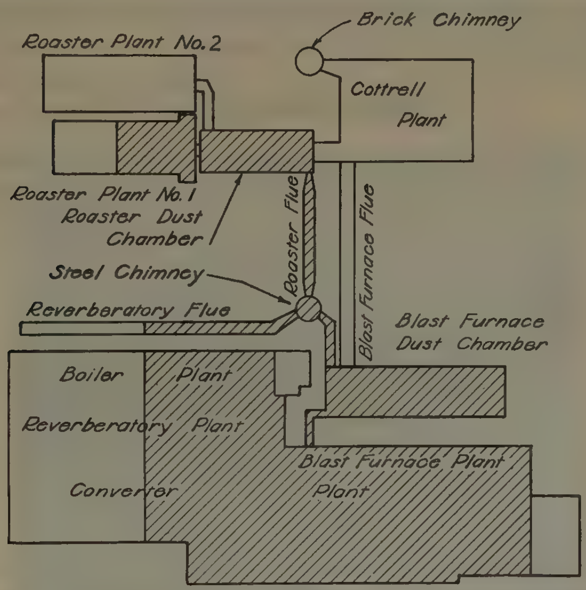


FIG. 1.—PLAN OF CHIMNEYS AND FLUE SYSTEM. ORIGINAL CONSTRUCTION CROSS-HATCHED.

were placed in operation in 1922. All roaster, blast-furnace and converter gases pass through the treater to the new brick chimney, the reverberatory gases going to the old steel chimney as formerly. Essentials of the plant layout are shown in Fig. 1.

COTTRELL CHIMNEY

The Cottrell chimney, 29 ft. top internal diameter by 430 ft. high, was built of radial perforated brick laid in a 1:1:3 cement-fireclay-sand mortar. The sand was local river sand, samples of which in 1931 showed about 11 to 13 per cent lime content. Crushing tests on the perforated brick laid in this mortar showed an average compressive breaking stress of 1207 lb. per sq. in. of gross outside area, between the limits of 1100 and 1350 lb. Design calculations on the chimney give 317 lb. per sq. in. as the maximum compressive stress when calculated for a wind pressure

of 30 lb. per sq. ft. over one-half of the projected area. While the factor of safety is less than four to one, the assumed wind pressure is far above anything ever experienced locally.

A 4-in. lining of the same material as the stack shaft was built to the 100-ft. level. The lower 55 ft. is vertical, supported on the foundation, and the remainder is supported on corbels built into the chimney at intervals, and is carried parallel with the shell, with a 4-in. air space back of the lining. No lining was provided above the 100-ft. level.

The usual clean-out hopper at the bottom of the stack was incorporated in the design, equipped with suction pipe and fan for transporting accumulated dust to the roaster dust chamber.

Cottrell Chimney History

About 1926, fragments of brick were noticed in the accumulated dust. As a matter of fact, when the hopper was completely cleaned out during the present shut-down period, brick spalls were found practically

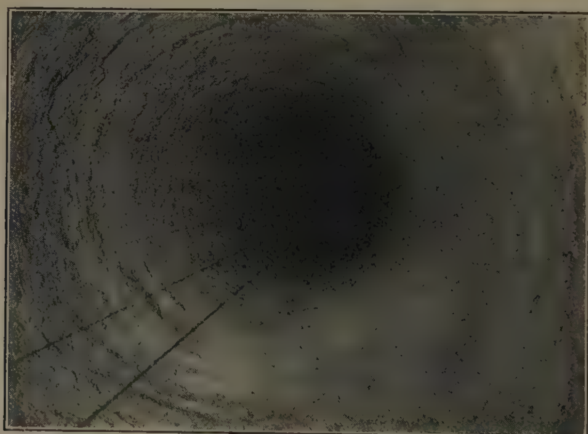


Fig. 2.—Looking downward from top of chimney. Extent of Spalling is noticeable.

at the bottom of the dust, indicating that spalling had started almost at the beginning of the operation of the stack. Over a period of about four years from 1927, brick spalls recovered amounted to approximately 130 tons, but as the dust hopper could not conveniently be cleaned out entirely, it was realized that this was not a true measure of the amount of spalling. Occasionally, whole 4-in. bricks were found, indicating either that considerable areas of the surface were letting go at once, or that a portion of the lining was coming down.

Careful examination of the entire outside surface in 1929 failed to show any cracks, and in 1930, for the first time, spalling of the inner surface was seen from the top (Fig. 2).

Condition of Chimney Interior

The plant was shut down in May, 1931, and advantage was immediately taken of the opportunity offered for a complete investigation of conditions. Roof-framing beams were first placed across the top, sufficiently heavy to support movable platforms or staging for permanent relining operations, and an inspection cage (Fig. 3) was provided for making a thorough examination of the interior surface of the chimney.

It was found that approximately 70 per cent of the inside area of the chimney above the lining was spalled off from 2 to 5 in. deep, with occasional small areas as much as 8 in. deep, and most of the remaining area was in serious condition. The lining was intact and there was practically no spalling below the 115-ft. level.

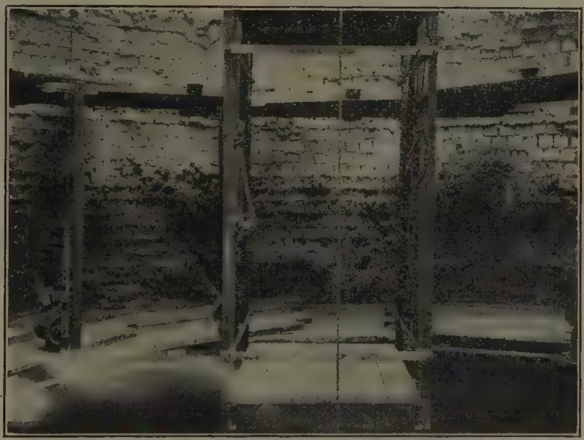


FIG. 3.—CAGE AND INSIDE SCAFFOLD. LATER THE CAGE WAS ENCLOSED.

The results of careful determination of acid and SO_3 in the brick and mortar at three different levels are shown in Fig. 4. In general these show some free acid penetration into the brick itself, in some cases as much as 4 or 5 in. Apparently this results in no harmful effect on the bricks, as there is no chemical reaction in a well burned brick. On the other hand, the mortar between the bricks and in its perforations reacts with the acid, forming sulfates, which absorb moisture from infiltrated air or from the gases and crystallize with an appreciable increase in volume. This increase in volume spalls off the inner surface of the brick, exposing new untouched areas of mortar, and the cycle is repeated. Our investigations did not conclusively indicate which sulfate was most responsible for the damage. Calcium, magnesium and aluminum were all suspected. One may visualize the tremendous force exerted by the hydration and crystallization of anhydrous sulfates in a confined space by the fact that 1 molecule of sulfate of calcium crystallizes with 2

molecules of water; 1 of magnesium with 7 of water; and 1 of aluminum with a maximum possibility of 18 of water.

Fig. 5 indicates the progressively increasing depth of spalling beginning at about elevation 115. Thickness of wall also is shown. Cursory examination of the chimney shaft behind the lining, below the 100-ft. level, indicated that no spalling had taken place. The conclusion is that temperatures of the lining at these levels have always been sufficiently high to prevent condensation of acid, and the wall thickness is sufficient to prevent infiltration of air. As both the wall thickness and

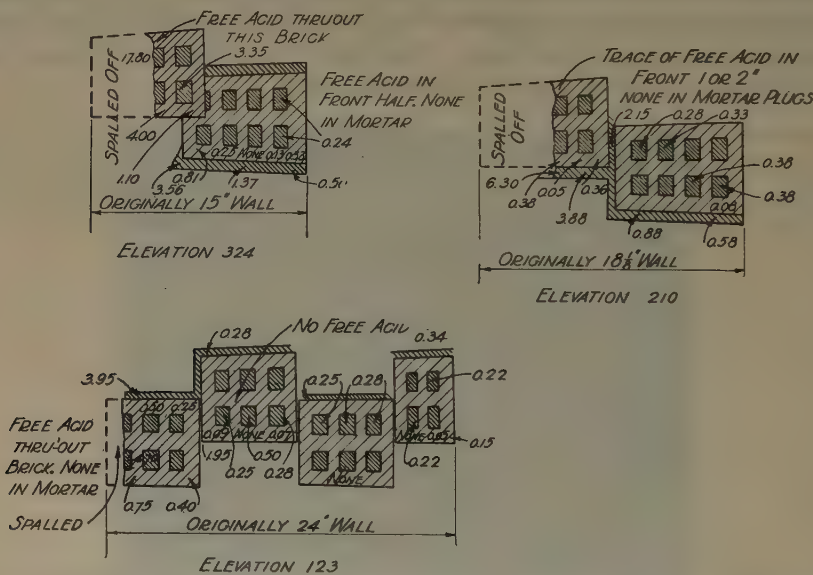


FIG. 4.—ACID AND SO_3 DETERMINATIONS IN BRICK AND MORTAR SAMPLES. FIGURES INDICATE PERCENTAGE OF SO_3.

gas temperatures decrease with increasing height of chimney, condensation of acid and crystallization of the sulfates occurred in the mortar joints and plugs and promoted the disintegration of the brick as outlined above.

Reference has been made in an earlier paragraph to the excellent present condition of the original steel chimney, built in 1915. Undoubtedly this is due to the fact that during operations from 1915 to 1922, when it was carrying all the gases, the plant as a whole was predominantly a blast-furnace smelting plant, and the roaster gas was neutralized by the other gases, particularly by the dust content. The dust contained appreciable zinc oxide. Furthermore, untreated roaster gases are not as corrosive as gases that have been treated by the Cottrell process, which removes the dust and not the acid. An additional feature is the absolute exclusion of moisture-laden air by the steel shell.

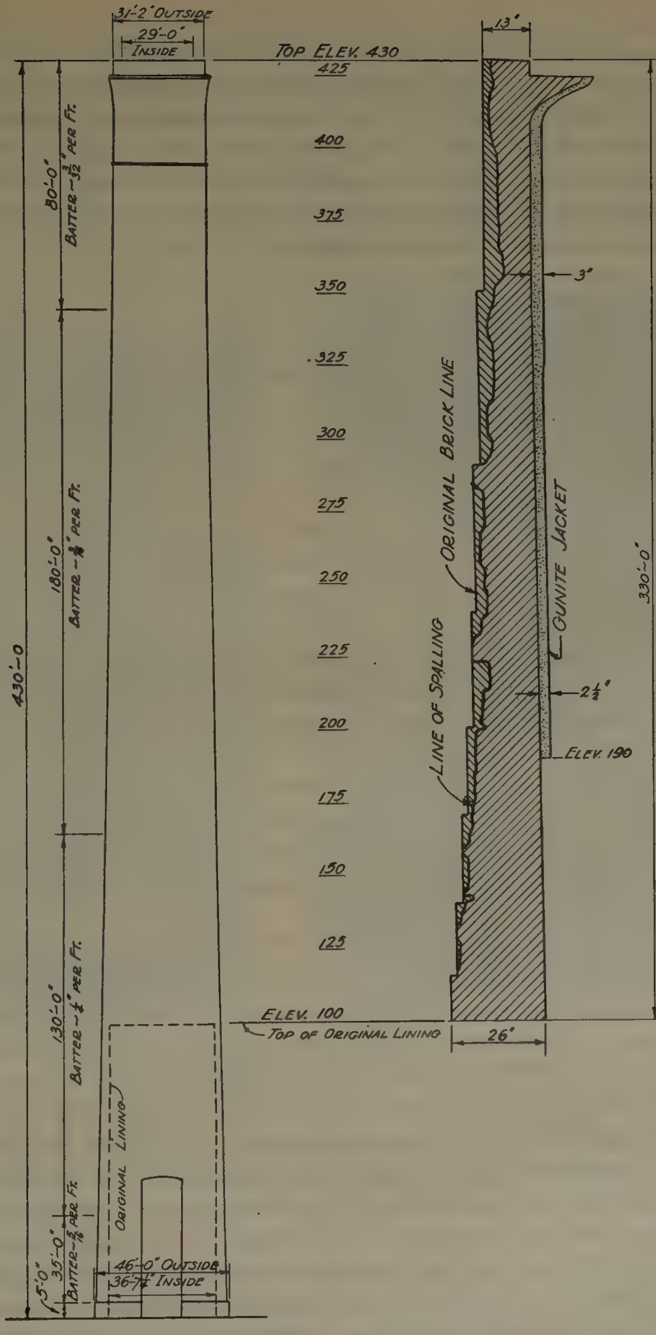


FIG. 5.—TYPICAL SECTION OF CHIMNEY WALL ABOVE ELEVATION 100 (DISTORTED SCALE).

Rehabilitation of the Chimney

The structure was rapidly approaching a dangerous condition. The top 80 ft. of wall, originally 13 in. thick, was practically one-third gone; in some small areas the brick remaining was only 7 in. thick. Spalling was progressing more and more rapidly during the shutdown, and from a consideration of the conditions and conclusions as to their cause, the problem resolved itself into the following:

1. To restore the upper portion of the chimney at least to its original structural strength.
   (*a*) By tearing down and rebuilding.
   (*b*) By some method of reinforcement.
2. To prevent the infiltration of air by the application of some protective, impervious coating.
3. To restore the inner surface of the chimney with an acidproof material.

It was decided that objectives 1 and 2 could best be accomplished by incasing the chimney in a reinforced concrete jacket applied by guniting; the interior work to consist of a complete veneer lining above the 100-ft. level of hard-burned solid brick laid in acidproof mortar.

The application of an acidproof veneer bonded into the stack structure is a new departure in American practice. The usual method, where a lining is included, is to provide an air space and support the lining on corbels.

A thorough investigation of the brick ultimately adopted included a determination of its mean coefficient of expansion. This was found to be 4.13×10^{-6} between 100° F. and 500° F., as compared with 8.33×10^{-6} for the brick of the original construction, and 5.5×10^{-6} for gunite.

The Gunite Jacket

This is 2½ in. thick from elevation 190 ft. to 324 ft. and 3 in. thick the remainder of the height. It was applied in two courses, the finish course being ½ in. thick. Reinforcing consists of welded steel rectangular mesh No. 0 wires (approximately 5/16 in. diameter) spaced 3 in. on center and No. 1 wire (approximately ¼ in. diameter) 8 in. on center. The fabric was delivered to the site in sheets 5 ft. 9 in. by 24 ft. 8 in., the heavy wire laid the length of the sheet. As applied, a complete band, one sheet wide, was placed around the chimney, the sheets being lapped two vertical bars and wired every 3 in. with No. 10 soft wire laid double. The reinforcing was stretched as tight as possible in wiring together, and after the complete circle had been connected up, wooden wedges were driven against the surface of the chimney to hold the mesh at a uniform distance of ¾ in. from the brick. These wedges were removed as guniting progressed. Fig. 6 shows the gunite rigging and Fig. 7 the

appearance of the mesh in place before guniting. Working from a two-stage scaffold, the reinforcing gang applied the steel course ahead of the gunite crew, each course being wired to the preceding course on 16-in. centers; i.e., every second vertical wire.

Proceeding from the 190-ft. level, the jacket advanced to the top of the chimney at the rate of approximately one 5-ft. 9-in. course per day. Then, working downward, the ½-in. top course was applied. Ring joints of the finish course were staggered from those in the base course,

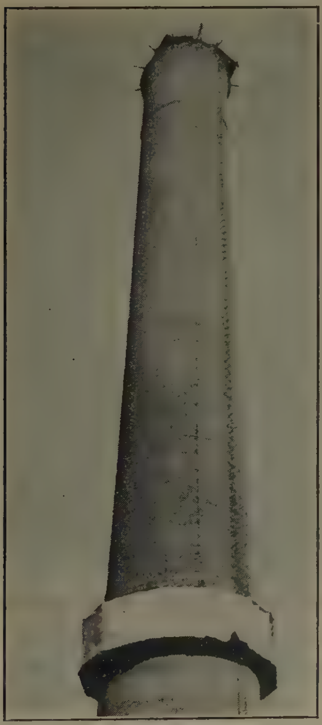


Fig. 6.

Fig. 7.

Fig. 6.—Gunite rigging at beginning of operations.
Fig. 7.—Gunite reinforcement at 190-ft. level.

to prevent shrinkage cracks opening through, and the fresh area was given a rough trowel finish as the work progressed. The entire surface was thoroughly wet down each morning. The top 55 ft. was painted with a heavy acid-resisting paint.

The mixture was 1:3 cement-sand, and the entire job was shot from one setting of the cement gun at the base of the chimney. Line pressure varied from 65 lb. for the 190-ft. level to 85 lb. at the top. The ground crew consisted of one gunman and two shovelers. The steel was handled by three men, one of whom worked only partly on the steel,

during raising and placing, and the remainder of the day served as helper to the nozzleman, removing wedges, etc., as the guniting progressed. One hoistman, an inspector and a superintendent completed the organization. The work was done under contract by the Western Gunite Co. of Los Angeles. The job was started in January and was delayed at various times when temperatures dropped below 35° F., the lower limit prescribed in the specifications. The completed jacket is shown in Fig. 8.

The Acidproof Lining

On completion of the gunite jacket, the rigging was moved inside, and, beginning at the top, a rough cut was taken over the entire surface, removing accretions, dust, and all loose and slabby brick areas (Figs. 9 and 10). Three men, with chipping hammers, constituted the working force, the job being completed in 22 days.

FIG. 8.—THE TWO STACKS, AUGUST, 1933.

Brick for the new lining is a hard-burned, clay brick, thoroughly vitrified, supplied by the Trinidad Brick & Tile Co., Trinidad, Colorado. Competitive tests were made on samples submitted by various manufactures, and some excellent materials were offered. Crushing tests of kiln-run samples on completion of burning for this contract showed an average compressive breaking stress of 11,000 lb. per sq. in. between the limits of 6,400 and 18,000 lb. They are practically impervious to acid. It was decided to lay the bricks to make a 3-in. thick wall lining, and accordingly most of them were supplied in a radial 3 by 4 by 8½ in. long with radial cuts of about ⅛ in. across the 3-in. face. These bricks were repressed in a special mold. So-called "header" bricks 3 by 4 by 6½ in. were provided for anchorage headers into the old wall, and a few standard 2¼ by 4 by 8½ in. were provided for backing and fitting.

These bricks were laid in an acidproof mortar developed and tested in the laboratories of the United Verde Copper Co. Briquets under test showed remarkably high tensile and compressive stresses. Test panels of the brick and mortar were built in the Cottrell outlet flue and subjected to severe tests simulating conditions prevailing in operation.

These tests were met successfully. The mortar as developed is capable of close control as to time required for "set." This period is influenced by temperature and humidity, and it would have been impossible to obtain satisfactory results without controlling the mortar set as conditions varied.

In addition to the mortar for laying brick, it was necessary to provide a backing material for filling the irregular spaces between the new and old brick, especially where the old surface was broken and irregular.

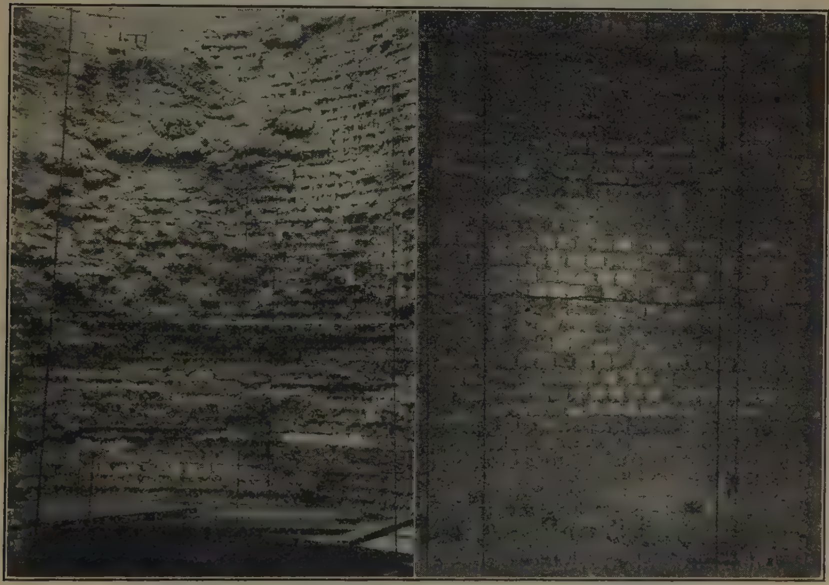


FIG. 9. FIG. 10.

FIG. 9.—TOP SECTION OF WALL BEFORE CLEANING. ACCRETIONS DID NOT EXTEND BELOW 400-FT. LEVEL.

FIG. 10.—WALL AT ABOUT 320-FT. LEVEL AFTER FIRST PRELIMINARY CLEANING. The four-course bands of smooth brick show where a full brick has disappeared. Intermediate bands spalled off to practically uniform surface.

This backing material was made of crushed silica ($-3/8$-in. mesh $+ 40$ mesh), mixed with sodium silicate solution and a setting accelerator. It is acidproof and of satisfactory density when properly screened, graded and mixed.

Beginning at the 100-ft. level, a full header course was laid, properly bedded and backed, as a foundation (Fig. 11). The 8½-in. radial bricks were used for this course in order to get the best possible anchorage. As mentioned before, there was no spalling at this level, and special effort was made to secure a firm bond to the old brick surface. Accordingly, another full header course was cut in, 3 ft. higher. Above this point, header courses were not cut in the full circle; generally 35 to 50 per cent or more, depending on conditions (Fig. 12). Fig. 13 shows

the method of distribution of these "anchorages," and Fig. 14 the finished lining.

Cutting for header courses and trimming the wall to line and surface was accomplished best by working the cutting gang on the day shift, the bricklayers coming on for the evening shift. Thus an interval of

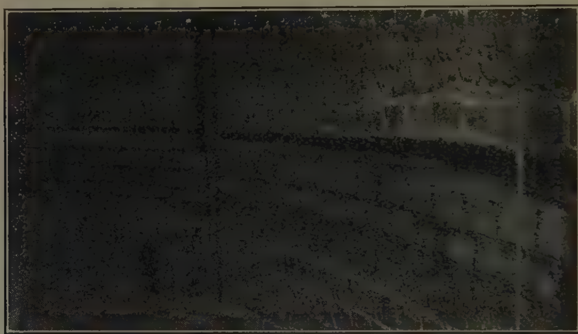


FIG. 11.—FULL CIRCLE HEADER COURSE CHANNEL AT 100-FT. LEVEL. THREE COURSES OF NEW BRICK ON TOP OF OLD LINING.

8 hr. elapsed after the bricklayers finished their shift before the work was disturbed by the vibration of the air hammers. This was ample time for a complete set of the mortar and backing. Header channels were cut one brick course above the top of the previous shift's work, thus minimizing unequal settlement of the fresh work. It was found that

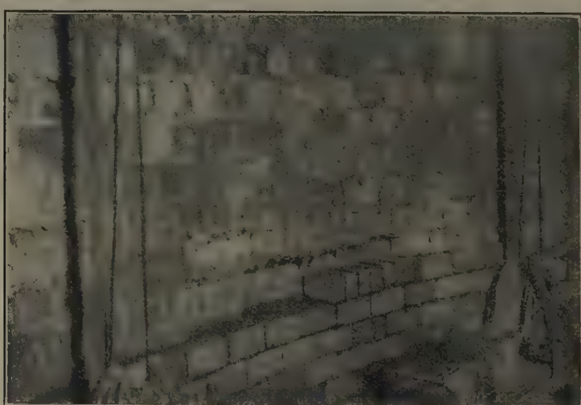


FIG. 12.—INTERMEDIATE HEADER COURSE AT ABOUT 300-FT. LEVEL.

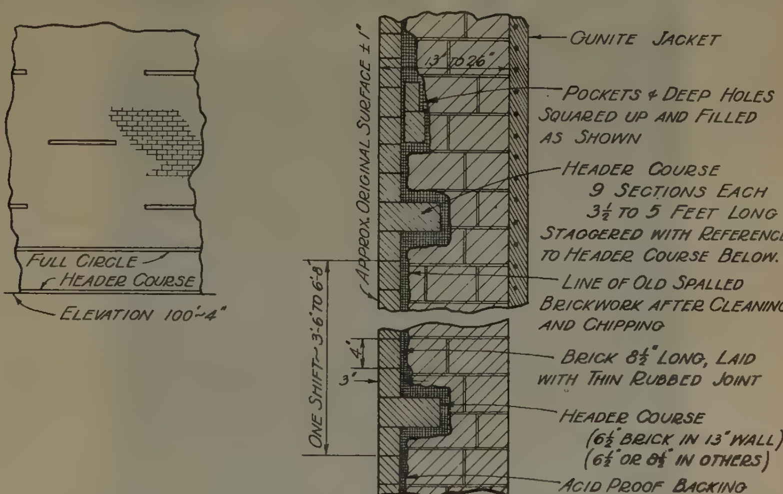
mortar joints in the existing brickwork had to be disregarded in locating the headers, as the old brickwork was in some cases as much as 4 or 5 in. out of level in a given course.

Usually the cutting gang consisted of three men. They raised and serviced the scaffold, cut and cleaned the wall, and removed all dust

and debris from the platform. The brick gang consisted of three bricklayers, two helpers on the scaffold, one brick passer on the ground, two mortarmen, and one hoistman. This work was accomplished in 77 days, a total of 868 man-shifts being required. Material quantities were as follows: 153,000 (600 tons) bricks; 73 tons mortar; 209 tons backing.

Equipment

The surfacing of approximately 25,000 sq. ft. of the outside and 33,000 sq. ft. of the inside of a chimney, from 100 to 430 ft. above ground,



TYPICAL SECTION OF FINISHED LINING

Fig. 13.—Part of developed surface showing location and arrangement of header (anchorage) courses

called for the assembly and handling of equipment not usually required by a smelter-operating force.

After the first steel was in place at the top of the stack (the erection of which was contracted), a 12-arm spider was erected, from which to hang the suspended scaffold, both inside and out. Twelve pairs of Patent Scaffolding Company's ratchet drums were leased from the owners, each drum wound with 200 ft. of ⅜-in. plow-steel cable. These ratchet drums are placed at the scaffold, and by reeling in the cable, the scaffold is raised. Accordingly, additional cable was spliced to each of the 200-ft. cables and attached to the spider outriggers, with the drums and platform putlogs hanging at the proper level. The scaffold around the stack was completed with 2 by 12-in. plank. The lower deck, as shown in Fig. 7, was hung 8 ft. below the upper. On account of the high wind prevailing at times, this was completely enclosed with canvas to protect the gunite. Hoisting of men and materials for the gunite job was in an open bucket or sling, one of the outriggers being extended to

support a 20-in. head sheave. The hoist was an 8 by 12-in. single-drum mine hoist, operated by air.

When the rigging was moved inside, the same method was used, except that the ratchet drums had to be on the lower platform, the upper being provided for protection against falling brick and debris, and supported on angle-iron frames. A single-deck cage 3 ft. by 4 ft. 6 in. was swung, using 1-in. mine-hoist cables for guides. The head sheave blocks

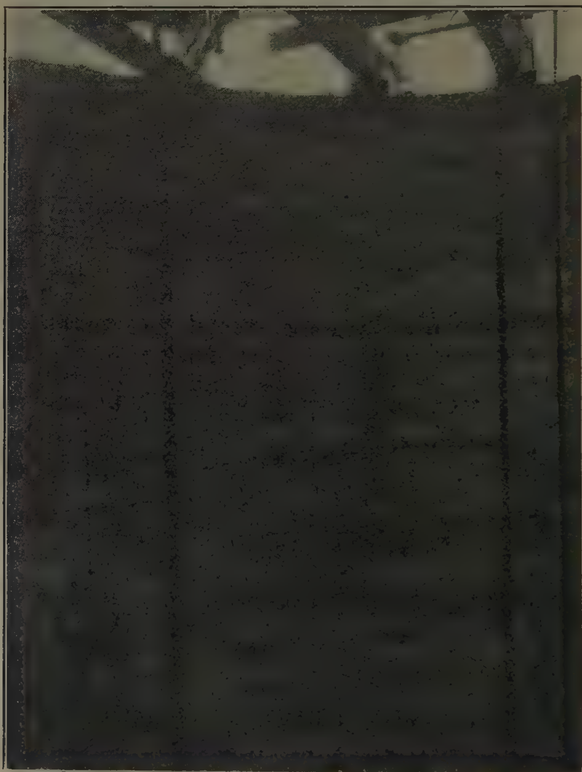


FIG. 14.—APPEARANCE OF FINISHED LINING AT TOP.

were mounted on a movable frame to compensate for the increasing diameter of the stack at the lower levels and bring the cage out to the scaffold. The scaffold framing was provided with "expanders" to hold it out to the wall line. An electric signal provided communication to the hoist operator, and a telephone with loudspeaker attachment was used to tell the men in charge of material what was needed.

Brick was handled by gravity roller conveyor from stockpile to within 30 ft. of the cage. Carrying trays holding eight of the 3 by 4 by 8½-in. bricks were loaded and piled into the cage by the men on the ground and removed by the helpers at the scaffold. Mortar was mixed by hand

and transported in trays holding about 60 lb. Backing material was mixed in a 4-cu. ft. rotary air-driven mixer and handled in 125-lb. trays.

Excepting only the erection of the roof steel framing and the gunite contract, all labor of rigging and construction was handled by the smelter personnel. There were no serious injuries.

While the Cottrell plant has not been operated since the repairs were completed, it is felt that the chimney at present offers considerably longer life than as originally built. The total cost of repair was approximately 40 per cent of that of a new acidproof chimney.

Recovery of Suspended Solids from Furnace Gases in Copper Smelters, with Special Reference to the Cottrell Process of Electrical Precipitation

By Harry V. Welch,* Los Angeles, Calif.

This paper presents a brief discussion of numerous devices and processes which have been utilized for the recovery of values from gases in copper reduction works and describes in greater detail the development and application in recent years of electrical precipitation, more familiarly known as the Cottrell process, to the problems of dust losses with furnace gases in copper-smelting and refining plants.

The field for collection of suspended particles in gases is widely extended in our modern complex, congested and industrialized mode of living, and its satisfactory solution is required by our increasing appreciation of the extravagant waste, unhealthful effects and nuisance which such uncontrolled emission of solid suspensions represents. Much of the special dust-recovery equipment in use today has been developed in one industry and carried over to many others. The early work of Cottrell was with sulfuric acid mists, aimed at the purification problem in contact acid manufacture, but the present uses of Cottrell equipment cover a diversified field—carbon black, combustible gas, oil mists, iron blast-furnace fumes and dusts, cement-kiln gases, as well as the purification of gases in contact and chamber acid plants and its wide and extended use in the nonferrous metallurgical field of copper, zinc and lead, and their associated metals.

Dust and fume collection from furnace gases became a problem only with the modern development of large and amply powered auxiliary furnace equipment, which in turn permitted large furnace units, fast driving and large outputs of finished product per unit of furnace capacity. The ancient metallurgist with his small clay pot and hand-operated bellows had too little gas velocity to cause a real dust loss. As furnaces grew in size, flues and stacks were added, and undoubtedly the first observable dust losses were indicated by the deposits in the flue and stack base. Large flues, it was noted, collected more than smaller ones, and the expansion of a portion of the flue into a chamber represented the first serious attempt at recovery of furnace dust losses. For a century or more the settling chamber represented the best of skilled metallurgical practice in recovery of furnace dust, and it may be said to have reached

* Chief Chemist, Western Precipitation Co.

its peak of good engineering practice in the dust chambers of the new Anaconda plant completed in 1902,[(6)]* and the extended and enlarged hillside settling flue erected some years later. The year 1906 saw the beginning of the fundamental investigation and experimental work of Dr. F. G. Cottrell on electrical precipitation, while the first commercial installation at a copper smelter went into operation in 1911, at the Balaklala smelter at Coram, California.

During this same five-year period many investigators were at work on the problem of stack loss at copper smelters and two large plants involving very different ideas were placed in operation. It was during this same period that a very epidemic of complaint and litigation broke out in the areas surrounding our western smelters and the incidents of this litigation served to focus attention on stack losses, and to make available technical talent and finances for investigation and for installation of plants for the recovery of the dust losses.

At the Great Falls plant of the Anaconda Copper Co. an elaborate investigation of baffled dust-chamber construction was conducted, which led to the construction in 1909 of the company's wire-hung dust chamber. This plant and the results secured have been excellently described by Goodale and Klepinger.[(12)] Suffice it to say that on operations and tonnages then passing through the plant it made a recovery of the copper in the suspended solids approximating 95 per cent, and the total weight of collected material in something over three years' operation was upwards of 100,000 tons. Changes in material to plant, to decreased copper operations and corrosion problems due to works shutdowns, later failed to justify the maintenance cost of the wire system, and at present, as an adjunct to the zinc plant, it is operated as a simple unbaffled dust-settling chamber.

During the years 1909 to 1911, the two copper smelters in northern California, the Mammoth smelter of the United States Smelting, Refining & Mining Co., at Kennett, and the Balaklala smelter at Coram, were in conflict with the farmers of the northern portion of the Sacramento Valley. The Mammoth smelter met the decrees of the Court by the installation of an extensive cooling system and baghouse for its blast-furnace gases, while Balaklala installed the Cottrell process of electrical precipitation on the gases arising from the multiple-hearth roasters and copper blast furnaces.

A baghouse was feasible on the Mammoth plant gases since the furnace charge contained an appreciable percentage of zinc, sufficient of which was volatilized to neutralize the free SO_3 present in the gases, and thus preserve a reasonable life to the bags. C. T. Rice[(8)] has ably described this plant and provided figures of construction cost and operation. Since the baghouse dust approximating 23 per cent zinc (as zinc

* Figures in parentheses refer to the bibliography at the end of the paper.

sulfate and zinc oxide) was unsuitable for retreatment in the copper-plant furnaces it was accumulated as a waste product and the baghouse operation represented entirely a nuisance requirement.

During the period of operation of the Mammoth baghouse, and more particularly during 1911, the Balaklala smelter at Coram, four miles down the valley from Kennett, constructed and placed in operation the first commercial electrical precipitation installation at a copper smelter.(7) This plant will be more particularly described later in the paper.

Early Experiments on Electrical Precipitation

The phenomenon of precipitation of suspended fume and dust particles under the influence of unidirectional high potential discharge and field was early noted in our modern development of science. Hohlfeld,(1) in Germany, in 1824 noted that the discharge from a Leyden jar through a wire centered in a bottle filled with smoke would effect the precipitation of the suspended particles. Since industry as we know it today, and even more so the translation of scientific discoveries to industrial application, was nonexistent then, the experiment passed as an interesting effect only and was soon forgotten. Again 20 years later, Guitard, (2) in England, noted the same phenomenon, and since the generation of electrical high-potential charges was still limited to frictional effects on rods of glass or wax, the discovery was soon forgotten. In 1876, however, Wimshurst brought out his electrostatic machine and thus made available vastly greater quantities of high-potential electrical energy, but still, as later events proved, entirely inadequate to successful operation in commercial electrical precipitation.

For the third time, in the early eighties the phenomenon of electrical precipitation of suspended solids by electrical stresses was discovered by Sir Oliver Lodge. He and his associates, making use of the Wimshurst machine, attempted the first commercial installation at the Dee Bank Lead Works in 1885. Public attention undoubtedly was first directed to the principles and possibilities of electrical precipitation by Sir Oliver Lodge's paper(4) of Nov. 3, 1884, at the meeting of the Liverpool section of the Society of Chemical Industry; and industrial attention by W. M. Hutchings,(5) works manager at the Dee Bank Lead Works, who published a brief description of the plant, and by A. O. Walker,(3) who, during 1884 and 1885, secured English, American and Continental patents. The plant utilized as a source of electrical energy a Wimshurst machine consisting of two 5-ft. plates, driven by a 1-hp. steam engine. The electrode system installed in the outlet flue consisted of a H-shaped rod frame with a number of short point projections attached to the rod framework.

No further information on this plant appears in the literature; undoubtedly the results secured were unsatisfactory, as would be indi-

cated by laboratory tests with a Wimshurst machine, which showed that the output from a six-plate modern machine delivered too little electrical energy even for a laboratory-size section of pipe electrode. The energy requirement for satisfactory electrical precipitation is of appreciable magnitude, and it was the realization of this requirement, which can be met only by modern electromagnetic machines, that enabled Cottrell to bring to a successful issue the commercial process of electrical precipitation.

During the first decade of this century, conditions were opportune in several countries for this development. In fact, the three most industrialized countries, Germany, Great Britain and the United States, each had a scientific investigator at work, who achieved results that led to commercial installations—in England, Sir Oliver Lodge, in Germany, Dr. Erwin Moeller, and in this country, Dr. Frederick G. Cottrell.

Development of Cottrell Process

In 1906, Dr. Cottrell, then Professor of Physical Chemistry at the University of California, interested himself in the problem of purification of sulfur-bearing gases for contact acid manufacture. His investigations first followed along the line of mechanical operation involving high-speed small-diameter centrifugal tubes, but led to indifferent success, since the problem of removing suspended mist particles, which are the carrier for arsenic poisoning of the catalytic mass, is extremely difficult by mechanical means. Becoming convinced that a possible solution lay along the line of mist removal by means of high-potential unidirectional discharge, he set up from exceedingly limited equipment, a laboratory test unit that yielded a unidirectional potential of about 3700 volts. With this limited voltage and bare iron electrodes, effective mist precipitation was not secured, but Cottrell discovered that by covering his discharge electrode with a fibrous material such as asbestos, effective and visible electrical input into the gases could be obtained and mist precipitation effected. An excellent description of the early work appears in several articles by Cottrell,[7,9,15] Schmidt,[10,16] Bradley[11] and others, which will not be repeated here in detail.

In theory, electrical precipitation requires two definite electrical considerations to be fulfilled, with respect to gases passing between the electrodes: (1) an effective flow of ionizing current in one direction only from one electrode; (2) the maintenance of a suitable high-potential static field between the two electrodes. The flow of ionizing current is the means of electrically charging the suspended particles and may be considered as follows: One electrode, the discharge electrode, preferably negative, must possess a radius of curvature that is small as compared to the collecting electrode. It must be of such diameter that, under the static voltage that can be maintained without serious arcing, the potential gradient at its surface is sufficient to disrupt the gas particle's neutral

electrical condition, giving to it, as an end result, an electric charge of the same character as the discharge electrode. Such charged gas particle, under the existing static field, is immediately repelled by the discharge electrode and attracted by the collecting electrode, and therefore starts immediately to migrate across the field. In its movement of migration it collides repeatedly with other gas particles and also with the suspended dust particles, in turn charging them. The disruption and charging of the gas particles involves an actual change of energy levels, one visual effect of which, as a charged gas returns from a higher to a lower energy level, is the corona glow. As the corona glow is indicative of an electrical charging of the gas stream, the observation is of practical importance as a guide and requirement for commercial plant operation. It was this realization of the practical importance of the corona discharge that enabled Cottrell to make his rapid advance in the early days of commercial development. Not having at his disposal either in laboratory (4000 volts) or field (20,000 to 30,000 volts) the high voltage equipment of today (60,000 to 100,000 volts), Cottrell used asbestos fiber or mica edges to secure the diminution of radius of curvature of discharge electrode necessary with the potential gradient available in his equipment, to secure a visible corona glow and ionizing effect.

The ionized gas particles produced in the immediate vicinity of the discharge electrode are immediately subjected to the stress of the electrostatic field between electrodes, and as it moves forward it drags along mechanically a mass of gas, thus giving a windage effect designated as "electric wind" and familiarly known in the extinguishing of a candle light when placed between high-potential terminals undergoing sparking or arcing. As such electric windage must go in a closed circuit there result points or small areas of intense ionization on the discharge electrode, which disclose themselves by small brightly colored spots and larger areas showing only the corona glow.

While precipitation effects are secured with the discharge electrode, either negative or positive, experience has shown on good conductive gases and deposits, that with negative discharge, from one-fourth to one-third higher voltage can be maintained between electrodes than is possible with positive discharge. Since maximum efficiency of a precipitator is secured with maximum potential obtained without serious arcing, commercial installations are always operated with discharge electrodes negative.

It is also possible to get a precipitation effect with high-potential alternating current instead of the usual rectified supply, partly because in the usual form of precipitator of fine wire and plate or pipe, some rectification takes place; and partly because the agglomerating effect of an alternating current discharge provides suspended material more readily taken out by baffle and wall action and by settling. Substantially,

however, high-potential alternating discharge provides an agglomerating effect, coalescing a great number of individual particles into a loosely agglomerated mass in dry materials, and from mist to droplets in liquid suspensions. Commercially, it is of little or limited importance.

While with mist and fume particles suspended in a gas stream of high humidity, investigation has disclosed that the length of discharge electrode per unit of gas volume per unit of time is a precise measure of unit capacity or of unit efficiency, it is quite otherwise when gases are less conductive and deposits on the collecting electrode are more or less nonconductive. Here the rate of discharge of the deposit often becomes the most significant and controlling factor of capacity.

Conductivity of Deposit and Conditioning of Gases

As distinct from a deposit of good conductivity, there are to be distinguished two general types of deposits: (1) the less frequent, a smooth-surfaced type of deposit such as pure zinc oxide, where as the deposit builds up efficiency decreases, and on the primary instruments at the switchboard current decreases and voltage increases. The result is equivalent to the insertion of a continuous liner of insulating material, and of course, adversely affects precipitation. (2) The second type is very common, and here the built-up deposit acts as a discontinuous dielectric or porous dielectric. At the points or edges of the discontinuity of the dielectric deposit, sufficient electrical stress develops to ionize the gas, and therefrom a reverse flow of current takes place between electrodes, with the result that the electrostatic voltage which can be maintained is greatly decreased, primary amperes readily exceed the transformer rating, primary voltage falls, and precipitation efficiency rapidly decreases.

As a corrective means, so-called "conditioning" of the gases and the collecting electrode deposit is provided, and most commonly this is accomplished by water sprays introduced into the gas stream previous to its entrance into the precipitator. Water sprays are by no means the only conditioning agents employed, for a vastly smaller quantity proportionally of sulfuric acid fumed into the gas stream will effect the same result, and for conditioning deposits, oil and tar mists, graphitic and carbon dusts, iron-ore dust and the like have all been found beneficial and possible of application under specialized conditions existing at some industrial plants. However, water commonly forms the cheapest means of conditioning gases and deposits and its use where basic suspended particles such as lead and zinc fumes, as discharged at some converter plants, are to be treated, is standard practice.

Conditioning of the gases when required is a relatively simple proceeding, requiring little water, and gives improved operation even at temperatures of several hundred degrees Centigrade. Copper-plant

multiple-hearth roasters commonly have available in their gases abundant acidity to condition the deposit, and for most localities the gas is also well conditioned, but certain plants in arid localities have noticed improved operation during periods of rainfall and the gradual decrease in this efficiency as the humidity lessened from day to day.

Requirements for conditioning a normally nonconducting dust or fume by water sprays are rather precise, and usually are met by the installation of a spray chamber where gas velocity can be decreased to from 6 to 9 ft. per second and wherein there are mounted a number of finely atomizing sprays. Such sprays usually are designed to deliver about ½ gal. of water per minute under a pressure head of 250 to 300 lb. As an alternative, an air-pressure atomizing spray using around 60 lb. air and water pressure and about 5 cu. ft. of free air per gallon of water, has been utilized at a small number of plants. A nonconducting dust is ordinarily made satisfactorily conductive if the collected dust contains from 2.5 to 4 per cent moisture.

It was stated that the loss of precipitation efficiency with a nonconducting or semiconducting suspended material, in comparison with material of good conductivity was largely due to the fact that the rate of discharge or rate of grounding of deposit became the controlling factor. It is evident, therefore, that by extending the electrode area sufficiently the equivalent in conditioning equipment may be obtained. For one Cottrell plant taking gases from copper converters and collecting a 50 per cent lead product, tests and calculations were established to provide this ratio. It developed that the equivalent of a suitable spray-tower conditioner would be met by adding in series to the already installed precipitator, from two and one half to three additional precipitators. This is not a fixed ratio, to be translated to other plants directly, as several factors enter into consideration, especially that of temperature. In this particular instance the gas temperature was about 200° C. and indications were that the ratio decreased with decreasing temperature, and increased with increasing temperature.

Considerations involving the handling of poorly conducting deposits have not, by many authors, heretofore been considered essentially a copper-plant problem. However, the majority of copper plants do concentrate into their mattes, and discharge with their converter gases, decidedly commercial amounts of lead and zinc, and with the strong probability of a low price for copper in the future, this recovery and marketing of such byproducts becomes of significant importance.

Electrical Precipitation Plants

In the following descriptions of electrical precipitation plants in copper works it is evident that the Cottrell byproduct of lead in some plants has practical importance.

There is no important copper smelter in the United States or Canada today without its complement of a Cottrell department-process and apparatus. For more detailed descriptions of many of these plants the reader is referred to the selected bibliography at the end of this paper.

Where operating figures are offered here it is to be understood that they do not represent practice of today (1933) but represent normal operations in normal times and with normal copper markets such as existed for several years previous to 1929.

Balaklala.—Balaklala, the first commercial precipitator erected, went into operation in 1911.[7,15] It consisted of nine Cottrell units in parallel, each unit equipped with a louver damper to inlet and outlet flue. Each

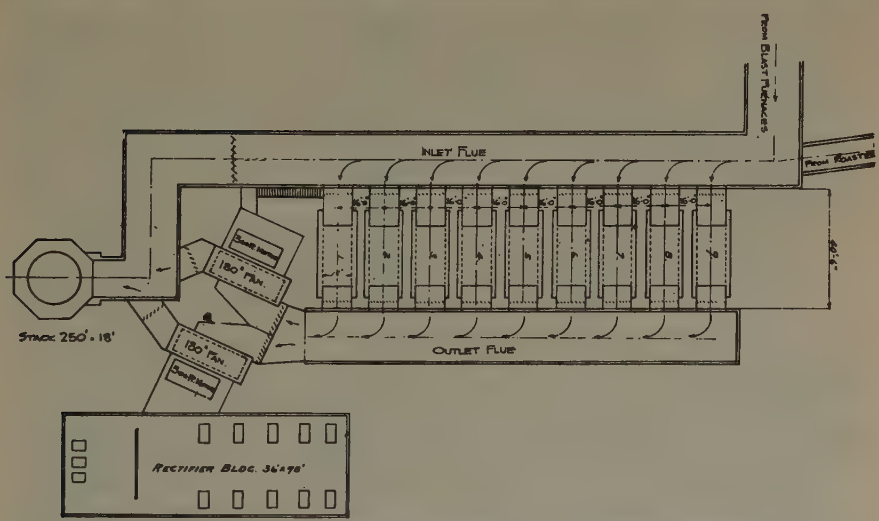


FIG. 1.—ORIGINAL FLUE ARRANGEMENT AT BALAKLALA SMELTER.

unit contained 24 ducts or gas passages, 5 in. wide, 24 ft. long, 10 ft. high. Collecting electrodes were strips of 10-gage plate 5 in. wide by 10 ft. long, alternating with discharge electrodes made of serrated mica strips held between twisted wires. To a limited extent, asbestos fiber between twisted wires was also utilized. Tension of the discharge electrodes was secured by springs. The potential across the electrodes varied from 20,000 to 30,000 volts. A bank of three power transformers transformed incoming 60-cycle, 2300-volts, three-phase power to three single-phase circuits at a variable voltage of 20,000 to 30,000. One double-bar synchronous motor-rectifier was directly connected to each unit of the precipitator. Gas volume varied from 200,000 to 300,000 cu. ft. per min. at 100° to 150° C. Efficiency on copper recovery averaged between 80 and 90 per cent. Plans of original flue arrangement and vertical section through one unit are shown in Figs. 1 and 2.

Garfield Smelter.—The Garfield smelter, at Garfield, Utah, when installed[9,13,15] in 1911 had 5-in. pipe test units. The present plant[36] on converter gases, with 260,000 cu. ft. per min. at 115° C., consists of nine pipe units in parallel, each unit containing three hundred and sixty 5-in. pipes, 10 ft. long, connected to an individual 15-kva. synchronous motor set. The pipe precipitators are in series with a plate-type Cottrell of eight units. Each unit has five banks of plates in series, and each bank or section contains 18 plates (17 ducts or gas passages) of corrugated plates 9 ft. 5 in. wide and 10 ft. long. The spacing between plates is 6 in. Each duct of a section contains 24 vertical weighted discharge wires, and two banks are operated from one 15-kva. electrical set, except that the last two banks in each section are connected to one 15-kva. unit.

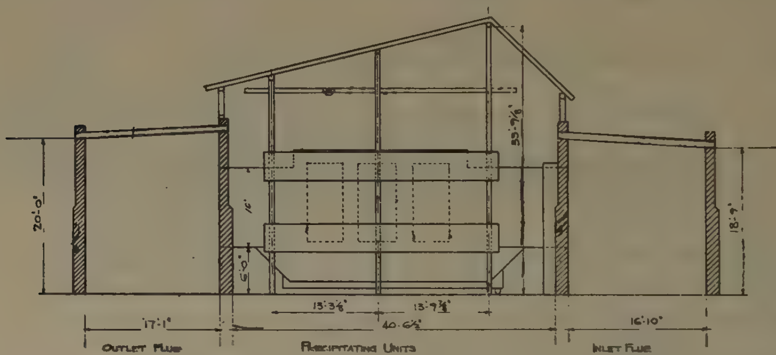


Fig. 2.—Vertical section through one Cottrell unit, Balaklala smelter.

Efficiency of collection at 19,000 to 20,000 volts across electrodes and for combined plate and pipe Cottrells is above 90 per cent for copper, lead, gold and silver. Since the collected product contains more than 50 per cent lead, the precipitate is essentially a fume, and it is to be noted that this collection is effected without humidification.

At the Garfield smelter, in addition to the large precipitator installation for converter gas there is also a large precipitator on the gases from the reverberatories and copper roasters,[36] which handles a gas volume of approximately 700,000 cu. ft. per min. at about 150° C., and consists of a plate precipitator of seven units, in parallel, each unit of three sections or banks in series and each section of 21 ducts or 22 plates. The plates are 11 ft. by 9 ft. 5 in., of corrugated iron, with a spacing of 6 in., with 496 vertical weighted discharge wires per section or bank. Operating potential between electrodes is 20,000 volts. At normal operation the efficiency is above 90 per cent. The general construction features of the roaster-gas precipitator follow that of the plate precipitator on the converter gases. Rapping of collecting electrode plates is accomplished by stationary air hammers, and a clever automatic device records the mechanical handling of the plant for each of the three daily shifts.

Anaconda Copper Mining Co.—The progress of the art of collection of suspended values in copper-smelter gases during the past 30 years is best exemplified by the activities of the Anaconda Copper Mining Co. Before the building of the new Washoe works in 1902 the plants were without designed means of dust recovery.[6] At the new works across the valley from the old smelter, which went into operation in 1902, very large dust-settling chambers were provided for the blast-furnace plant (275 by 38 by 36 ft.), the roaster plant and the converters. Adjacent to each dust chamber was its individual stack.

A few years after the new smelter went into operation upwards of 4000 ft. of additional settling chamber and flue, with a 300 by 30-ft. stack

FIG. 3.—COTTRELL INSTALLATION AT ANACONDA.

at its hilltop end, was placed in service. Of this new flue, 1234 ft. was of a width of 60 ft. and 995 ft. of a width of 120 ft. Throughout this distance of over 2000 ft. the flue was hoppered.

During 1914 and 1915 much investigational work on electrical precipitation was carried on by the Anaconda staff. This work involved testing both plate and pipe precipitators, and with the latter diameters were carried up to 4 ft. and potentials to 220,000 volts.[14]

In 1918[21] Cottrells (Fig. 3) were installed at the stack end of the main flue system and a new stack 585 ft. high, 75 ft. inside diameter at base, and 60 ft. inside diameter at the top, was provided. This installation consisted of 20 units in parallel, each unit having one section of 21 plates (20 ducts or gas passages) spaced 12 in. center to center, and each plate being 24 ft. high and 20 ft. long of corrugated iron. Discharge

electrodes originally were of chain, but later were replaced by No. 14 wires placed vertically and tensioned by weights. Units of a similar type are also in service on the gases from the special reverberatory smelting collected dusts and also at the No. 2 roasting plant (in operation in 1916). Electric power to the precipitation units is supplied by 12 electrical sets, each set comprising a 75-kva. motor-generator-rectifier set connected to a 75-kva. maximum 100,000 volt transformer. Voltage across electrodes approximates 65,000, with daily dust collection of about 120 tons, from a gas volume of 2,225,000 cu. ft. per min. at 150° C. Efficiency of recovery approximates 95 per cent on copper, gold and silver, 90 per cent on total dust, 86 per cent on arsenic trioxide and 84 per cent on lead. This main treating plant at Anaconda handles the largest gas volume through a single precipitation installation of any plant so far constructed, and with the additional units at No. 2 roaster plant (four units), the special reverberatory plant handling arsenic-bearing flue dust, (six units, operating with two in series, on conditioned gases), and the individual units on the roasters at the zinc plant, constitutes the largest investment in precipitation equipment as well as the most extensive settling chamber and settling flues at any one plant.

At the Tooele plant of the International Smelting Co., a subsidiary of the Anaconda Copper Mining Co., a plate-type precipitator is installed on the multiple-hearth roaster gases.[22,23,31] It consists of two units, each unit of four sections of plates in series, each plate 10 by 10 ft. The plates are of corrugated iron spaced on 9-in. centers, there being 16 plates (15 ducts or gas passages) per section. Electrically, two sections are connected through a continuous system of horizontal discharge electrodes, the latter being ¼-in. standard iron pipe spaced about 6 in. apart. Electrical power is supplied by four 15-kva. sets, and operating with a gas volume of 135,000 cu. ft. per min. at 200° F., the copper collection efficiency approximates 97 per cent and the total dust collection 93 per cent. The daily collection is about 18 tons dust.

Tacoma Smelting Co.—At the plant of the Tacoma Smelting Co., Tacoma, Washington, a pipe and a plate precipitator are in service, the former on the reverberatory-furnace and roaster-plant gases and the latter on the converter-plant gases. The pipe-type precipitator consists of 18 units, each of 84 pipes, each pipe 12½ in. inside diameter by 15 ft. long. This precipitator is served by 10 motor-generator sets connected to ten 15-kva. transformers delivering approximately 50,000 volts to the precipitator units, one being kept as a spare. Originally this pipe precipitator handled the combined gases from reverberatory furnaces, roasters and converters, and under that condition of operation handled upwards of 600,000 cu. ft. per min. at a temperature of 180° C. Later the converter gases were segregated and passed to a plate-type precipitator consisting of two units in parallel, each unit of six banks or sections

of plates in series. Later still a third parallel unit was added. Each section or bank consists of 27 plates (26 ducts or gas passages), each plate 8 by 12 ft. of corrugated iron. Plates are spaced 6 in. center to center. Each of the 18 sections is served by a synchronous motor-rectifier set and a 15-kva. transformer. Discharge electrodes are of vertical wires and the high-potential framework is carried through lime-dust seals and supported outside the precipitator on porcelain insulators. The operating potential on the first section approximates 25,000 volts dropping somewhat from section to section in series, to about 20,000 on the last section. The material collected is largely lead and zinc sulfate and oxide fumes, and operation is at times with partial humidification and at other times without gas conditioning. The gas volume varies from 100,000 cu. ft. per min. with one converter in service, to 180,000 cu. ft. per min. with three converters in service, with an upper temperature range of about 200° C.

Types in Arizona.—Of the eight companies in Arizona that have installed precipitators, four have utilized the pipe type only, one the plate type, one the so-called rod-curtain type, one a special screen type of plates with the gases passing through the screen, and one containing units of pipe, plate and transverse rod curtain operating in parallel.

International Smelting Co.—The first plant erected was that at the Inspiration smelter of the International Smelting Co. at Inspiration, Ariz.,(16,18,19) which went into operation with the starting up of the smelter in 1915.

Two separate treating plants were erected; one on the gases from the converters, the second on the gases from the multiple-hearth roasters functioning as concentrate driers. The two installations were served by a common substation power house. Both precipitators utilized 12-in. by 16-ft. iron tubes as collecting electrodes, the construction being that known as the exposed-pipe type, in which the gases enter a header below the tubes, move up through the tubes and pass to the stack from a top header. The exterior surfaces of the pipes are exposed to the atmosphere throughout their length and rapping is effected by a system of hammers striking on lugs fastened to the external face of the pipes. The freedom from appreciable amounts of zinc or lead in the matte gave a converter gas of considerable acidity and both gases and deposit are good conductors. In the multiple-hearth driers, the vaporization of the water from the flotation concentrates gave equally satisfactory conditioning of gases and deposit.

The converter precipitator was divided into 12 units, all in parallel, each unit containing 64 tubes of 12 in. inside diameter and 16 ft. long. An iron chain, later replaced by No. 12 weighted iron wire, central to each tube formed the discharge electrode. The operating voltages were from 60,000 to 70,000 volts. The gas volume from three converters

averaged 270,000 cu. ft. per min. at 150° C., the collection efficiency being 98 per cent on copper, and the daily collection from 2 to 3 tons of dust containing 35 per cent copper.

The original drier-plant precipitator in 1915 treated the gases from five multiple-hearth furnaces. It consisted of 12 units of 20 tubes each, similar to those in the converter Cottrell, or 12 in. inside diameter and 16 ft. long. The prevailing voltage was around 70,000 volts, efficiency over 99 per cent, and daily dust collection around 5 tons averaging 25 per cent copper. In 1920 additional driers and a duplicate of the

FIG. 4.—ROASTER PLANT WITH COTTRELLS MOUNTED ABOVE WEDGE ROASTERS, SOUTHWEST METALS COMPANY.

precipitator were installed. The power substation contains ten 15-kva. motor-generator sets direct connected to 15-kva. 100,000-volt transformers, and serves both the drier and the converter Cottrells.

Southwest Metals Co.—In 1916 the Humboldt smelter of the Southwest Metals Co. (then Consolidated Arizona Smelting Co.) constructed a new roaster plant of four 22-ft. Wedge roasters.(20) Integral with this construction, and mounted directly above the roasters, a Cottrell precipitator of the so-called submerged-pipe type was installed (Fig. 4). Each roaster was served by a two-unit Cottrell, each unit consisting of 40 tubes of 12-in. diameter and 16 ft. long, or 80 tubes per roaster Cottrell. Two Cottrells are served by one 15-kva. power set, a third set being reserved as a spare. The voltage at the precipitators was about 65,000 and the individual stacks from the roasters were clean looking, indicating continuous efficiencies well over 90 per cent. A single efficiency test gave 96 per cent. The gas volume per roaster averaged 16,000 cu. ft. per min. at temperatures varying from 150° to 300° C. Collection

approximated from 6 to 7 per cent of furnace feed, the latter running from 400 to 500 tons per day for the four roasters.

Magma Copper Co.—In 1925 the Magma Copper Co. placed in operation its smelter at Superior, Ariz.[29] The gases from a battery of five Herreshoff furnaces pass to a four-unit precipitator, each unit of 144 pipes, 8 in. diameter and 12 ft. long. Power is supplied from three 15-kva. motor-generator rectifier sets to three 15-kva. high-voltage transformers. Two power sets are commonly in service, one set to two pipe units. Operating voltage approximates 45,000 volts with 100,000 cu. ft. of gas per min., the daily collection being 16 tons per day, approximately 4 per cent of furnace feed, and containing 15 to 18 per cent copper.

Copper Queen.—A similar type of precipitator was placed in operation at the Copper Queen plant of the Phelps Dodge Corporation in 1927. This submerged-pipe precipitator consisted of eight units each of 140 tubes 8 in. in diameter and 12 ft. long, a total of 1120 tubes, and received the gases from twelve 19-ft. diameter, 11-hearth Herreshoff roasters, handling from 200 to 250 tons of feed per day per roaster. The total gas volume was about 145,000 cu. ft. per min. at 650° F. The dust collection averaged something over 200 tons per day, and represented from 6 to 8 per cent of the furnace feed. The efficiency slightly exceeded 95 per cent on copper and just under this figure for total dust. Rapping was most effectively accomplished by air hammers operating against the lower edges of the pipes. The eight units were served by eight power units, each with synchronous-motor rectifier and 15-kva. transformers. The potential at the electrodes approximated 65,000 volts. This Cottrell is not now in service because the smelter is no longer in operation.

Calumet and Arizona Mining Co.—In 1924, the Calumet & Arizona Mining Co. at its Douglas smelter placed in operation a precipitator to take the gases from the multiple-hearth roasting plant. The precipitator was designed for a gas volume of 500,000 cu. ft. per min. at 500° F. It consists of five units of the plate type in parallel, each unit of four sections or banks of plates in series, each section of 25 plates (24 ducts or gas passages), each plate 8 ft. long and 10 ft. 8 in. high. Spacing between plates is 6 in. The discharge electrodes are horizontal $\frac{3}{8}$-in. square twisted rods. Electrical requirements are supplied with eleven 25-kva. synchronous-motor power sets, the maximum voltage rating for the transformers being 45,000 volts. Operating voltage approximates 35,000 volts. The efficiency of copper collection exceeds 95 per cent, and the total dust collected amounts to 3 per cent of the roaster feed.

American Smelting and Refining Co.—At the Hayden smelter of the American Smelting & Refining Co., in 1920, the original dust chamber was divided by three longitudinal partition walls to form flue sections for a precipitator of four units, operating all in parallel.[24] Each precipitation unit contained four sections of screen collecting electrodes, and each

section consisted of five pairs of grounded screens placed across the chamber at right angles to the gas flow; i. e., the gases flow through the screens. The two screens to a pair are placed 6 in. center to center, with a vertical system of wire discharge electrodes between them. There is a space of 12 in. between the screens of adjacent pairs. Each screen is 8 ft. 6 in. wide, and 12 ft. 6 in. high, enclosed in an angle-iron frame. The screen material is No. 8 iron wire, with meshes 2½ by 2½ in. The inlet temperature varies from 100° to 350° F., the gas volume from 12 multiple-hearth roasters approximates 190,000 cu. ft. per min., and the usual operating potential is 24,000 volts. As the acid content of the collected dust is unusually high (which results in an appreciable corrosion problem), owing largely to the unavoidable low operating temperature, the collection efficiency is above the average and is well over 95 per cent.

United Verde Copper Co.—Precipitators of the pipe, plate and chain-curtain (later replaced by a rod-curtain) types were installed at the United Verde Copper Company's smelter at Clarkdale, Ariz., in 1922, and handled the gases as mixed from 24 multiple-hearth roasters, three blast furnaces and five converters.[(34)]

The pipe precipitators have exposed pipes, and down draft and the plate precipitators are similar in construction to those at Tooele, with horizontal discharge rods and vertical corrugated plates, the gases passing horizontally between the plates; i. e., parallel flow type. Later these corrugated plates were replaced by ⅛-in. standard pipe curtains, and more recently in part by ⅜-in. rods. The third form of precipitator approximated the transverse screen type in service at the Hayden smelter, and originally consisted of a series of vertically hung chain collecting electrodes with the gases passing through the electrodes; i. e., transverse flow. Later the transverse chain collecting electrodes were replaced by rods of ⅛-in. standard pipe.

The United Verde mine ore at times carries as much zinc as copper, and as a result the converter and blast-furnace gases carry appreciable quantities of zinc fume and therefore the Cottrell deposit is not as readily conducting as at several other Arizona plants.

The pipe precipitator at this smelter is arranged in two units, each unit with an independent flue system connecting to the main inlet and outlet flues of the plant. Each unit flue system supplies gas to and receives gas from six pipe sections, and each pipe section consists of 120 tubes, 12-in. diameter by 16 ft. long; a total of 720 tubes per unit or 1440 tubes for the two units. Inlet and outlet cone dampers are provided on the individual sections, and large slide dampers for major plant control on the unit flue connections to the main inlet and outlet flues. Rapping is effected by an upward hammer blow against projections on the sides of the pipes. The gas volume passing to these pipe precipitators approximates 430,000 cu. ft. per min. at an outlet temperature of 346° F.

Collection during 1929 averaged 76.9 tons dust per day, averaging 7 to 8 per cent copper. Four motor-generator rectifier sets with 25-kva. transformers supply electrical service, commonly operated on the 62,500-volt transformer tap.

The plate precipitator consisted of three parallel units, each unit of three sections of plates in series, each section of 17 plates (16 ducts or gas passages) spaced 8 in. center to center. The original collecting electrode plates were of corrugated iron 12 ft. high and 14 ft. long, and a system of horizontal ⅜-in. square twisted-rod discharge electrodes, 17 per duct, was provided. The gas volume for the combined throughput of the three plate units averages 238,000 cu. ft. per min. at an outlet temperature of 347° F. Efficiencies of 92 per cent and above are secured

Fig. 5.—Interior of electrical control room for Cottrell plant, United Verde Copper Company.

on copper, the collection in 1929 averaging 28.6 tons of dust per day, with an electrode potential of 50,000 volts. As corrosion and warping of these large plates developed, they were gradually replaced by rod curtains, consisting of ⅛-in. standard iron pipes on 2-in. centers, and supported in a suitable framework.

The transverse flow precipitator also consists of three units in parallel, with each unit of three sections in series. Each section or bank contains 10 sets of electrodes in series, each electrode set consisting of two (a pair) vertically hung collecting-electrode transverse curtains of ⅛-in. standard iron pipe. Each collecting electrode consists of 42 vertical ⅛-in. standard pipes 15 ft. 6½ in. long, spaced 2¾ in. on centers. These two collecting electrodes to a set are spaced 8 in. apart and between them is placed a discharge-electrode frame holding horizontal ¼-in. square twisted discharge rods. The discharge electrodes are spaced on 6-in. centers vertically, there being 33 rods per frame. Between the adjacent grounded

screens of two sets of electrodes the spacing is 11¾ in. There are thus in the three sections in series a total of 30 electrode sets and for the three units a total of 90 discharge electrodes and 90 pairs of transverse screen-type curtain collecting electrodes. Each transverse flow unit is supplied with electrical energy from an individual motor-generator rectifier set and 25-kva. transformer, the high-voltage connections being to the 50,000-volt tap.

The over-all efficiency of dust collection for the combined plant of two units of pipes (12 sections), three parallel flow units (three banks in series) and three transverse flow units (three banks in series), during 1929 was for copper, 91.5 per cent; on dust the parallel flow units were 2 per cent higher, the transverse 1 per cent higher than on the pipe units. On copper collection the parallel-flow units were 2 per cent, the transverse-flow units 6 per cent higher than on the pipe units. The power house or Cottrell substation is equipped with 12 motor-generator rectifier sets, two sets being in reserve as spares (Fig. 5).

United Verde Extension Co.—A rod-curtain precipitator is also in service at the smelter of the United Verde Extension Co. at Clemenceau, Ariz. The gases discharged from six multiple-hearth roasters pass to a two-unit precipitator of the rod-curtain type. Each unit consists of two sections in series, each section of 13 rod-curtain plates (12 ducts or gas passages), each rod curtain being built up on series of ⅛-in. standard pipes on 1½-in. centers, 33 pipes forming a curtain, the equivalent of a plate 4 ft. 2 in. long by 11 ft. 6 in. high. The spacing between curtains is 6 in., the precipitator voltage is approximately 35,000 volts supplied from one 15-kva. Cottrell power set. The gas volume averages 70,000 cu. ft. per min. at an outlet temperature of 157° F. The collection is 7 to 9 tons per day of 9 per cent copper, with efficiencies of 85 to 90 per cent on copper.

Canada

The smelter installations built in Canada during the past six years have embodied the rod-curtain Cottrell. This form of electrical precipitator has found application in lead smelters as well as in the field of industrial plants such as cement, powdered-fuel ash and so forth.

In substance it is the replacement of a plate by a framework carrying a series of rods, commonly ⅛-in. standard iron pipe tubes, on 2-in. centers, so that while the rods are rigidly guided to form a flat grid, they are free to move vertically under the rapping action or expand freely under temperature changes, thus giving a collecting plate from which the dust may be removed with ease, and which is free from buckling effects and warping under temperature changes. It is also favorably designed for ease of installation and repairs and for replacement where corrosion has occurred. In the later plants the formerly used discharge wire with

its weight for tension has been replaced by square twisted ⅛-in. bars, without weights.

The rod curtains are commonly 8 ft. long (in the direction of gas flow) and carry forty-eight ⅛-in. standard pipes on 2-in. centers, and vary in length from 12 to 18 ft., and even to 24 feet.

Noranda Smelter.—At the Noranda smelter at Quebec, Canada, the gases from eight multiple-hearth Wedge roasters handling a feed of 250 to 300 tons per day per roaster pass into six Cottrell units, each unit consisting of three sections or banks of rod curtains in series.[33,35] Each section carries 16 rod curtains (15 ducts or gas passages), the rod curtains functioning as plates being 18 ft. high by 7 ft. 4 in. long, and made up of ⅛-in. pipes spaced 1½ in. apart. The discharge electrodes are

FIG. 6.—INTERIOR OF ROASTER COTTRELL INSTALLATION AT NORANDA MINES, LTD.

⅛-in. square twisted steel rods suspended at 8-in. intervals, between the collecting electrodes. Twelve 25-kva. synchronous-motor electrical sets are connected to the six Cottrell units, and the substation contains a thirteenth unit as a spare. The gas volume approximates 450,000 cu. ft. per min. at 900° F. at the inlet to the Cottrell. (Fig. 6.)

At the converter plant two similar but somewhat larger units (18 ducts per section) handle 200,000 cu. ft. of gas per min. at 300° to 400° F. At this gas rating both precipitators have an efficiency of 95 per cent on copper. More recently the converter gas volume has increased to 350,000 to 400,000 cu. ft. per min., and under this heavy overloading efficiency of dust recovery varies from 65 to 70 per cent.

International Nickel Co.—Similar units are in service at the plant of the International Nickel Co. at Copper Cliff, Ont.[37,38] (Fig. 7). This precipitator comprises seven units, three sections long, each unit of 18 ducts, taking the gases from 30 roasters. A test with

15 roasters providing a gas volume of approximately 400,000 cu. ft. of gas per min. at 620° F. and four units of the precipitator in service, gave a copper-nickel collection efficiency of 97 to 98 per cent. The Cottrell substation is provided with fifteen 25-kva. power units.

Hudson Bay Mining and Smelting Co.—At the Hudson Bay Mining and Smelting Co. smelter at Flin Flon, Manitoba, units similar to those at Noranda are installed in three Cottrell plants, all served from a com-

FIG. 7.—COTTRELL INSTALLATION, INTERNATIONAL NICKEL COMPANY.

mon power house. Two units three sections long, 11 ducts per section, are on the gases from the multiple-hearth copper roasters, two units of the same size are on the gases from the multiple-hearth zinc roasters, and two units three sections long, with 15 ducts per section, are on the converter gases. In the latter, construction has been provided to permit ready expansion by addition of two more sections in series to the three installed, if and when the zinc-fume discharge from the converters lowers the conductivity of the dust and gases, or if a higher efficiency is desired.

South America

In South America Cottrell precipitation plants are installed at the Braden Copper Co. plant, at Caletones, Chile,[32] the Andes Copper Co.

plant at Potrerillos, Chile,[39] and the Cerro de Pasco Copper Corporation plant at Oroya, Peru. The newest unit at the latter plant utilizes the rod-curtain type of construction with rod curtains 24 ft. high.

Cerro de Pasco Copper Corporation.—At the plant of the Cerro de Pasco Copper Corporation, a new precipitator functioning as an arsenic (crude As_2O_3) collector was placed in operation in 1932 (Fig. 8). This new precipitator receives the gases after they have passed through the hot precipitators at the roaster plant and have been cooled to a temperature of approximately 120° C. It consists of three units, each unit three sections long, each section of 16 rod curtains (15 ducts), each curtain being 8 ft. wide and 24 ft. long. Rods are of ⅛-in. standard pipe spaced

FIG. 8.—CERRO DE PASCO COPPER CORPORATION'S 24-FT. ROD-CURTAIN UNITS.

on 1¼-in. centers. The spacing between curtains is 8 in., and the discharge electrodes, vertically placed, are of ⅛-in. square twisted bar. The precipitator handles a gas volume of 300,000 cu. ft. per min. at 120° C. with efficiency on dust and fume collection of 97 per cent. Ten tons of crude arsenic-bearing dust is collected daily. The high-potential electrical supply is provided by 10 sets of 25-kva. capacity, one set serving as a spare, and both altitude and the characteristics of the deposit tend to reduce the maximum operating potential to about 35,000 volts.

Electrolytic Refining

Since the copper smelters of the West as well as those in Canada and South America treat ores containing gold and silver values, electrolytic refining of the converter product is required. From the furnacing of the refinery products gold and silver values are discharged with the exit

furnace gases. Practice for the recovery on these furnace gases commonly provides a scrubbing, followed by a lead-pipe precipitator. The latest precipitators for this service are the plants at El Paso refinery of the Nichols Copper Co. and at the Montreal East refinery of the Canadian Copper Refiners, Limited.[40]

The former was placed in operation in 1930 and consists of two 30-tube lead-pipe units. Four pipes are 11 in. in diameter by 12 ft. 6 in. long, and 26 pipes are 8-in. diameter by 12 ft. long. The gas volume is 4000 cu. ft. per min. at 150° F. with an efficiency on gold and silver recovery of 98 per cent. Framework and supports are of steel, lead-covered where in contact with the gases, and supporting the sheet-lead work of gas inlet and outlet headers, and the lead electrode system. The discharge electrodes are of star-shaped lead-covered iron wire, supplied with a high-potential current from one 15-kva. Cottrell power set.

The plant at the Montreal East refinery* takes the gases from two doré furnaces and one Herreshoff slime roaster, and after the gases have passed through settling-flue chamber and scrubber they enter a lead-pipe precipitator consisting of two units of 22 p pes each. In each unit there are 20 pipes 8 in. in diameter and 12 ft. long, and two pipes 11 in. in diameter by 12 ft. 6 in. long. The gas volume approximates 3200 cu. ft. per min. at a temperature of 125° F., and electrode voltage of about 50,000 is provided from one 15-kva. synchronous-motor power set. The efficiency for gold and silver collection is on the order of 99 per cent.

APPENDIX.—Notes on Mathematical Formulas of Suspended Dust Separation; with Diagrams and Plots; also Sulfur Problems in Metallurgy

Diagrammatically, a Cottrell process precipitator is shown in Fig. 9. The nomenclature commonly used in discussing electrical precipitation is illustrated in Fig. 10.

Anderson[30] submits equations for the gravitational effect in settling chambers, effect of inertia in baffled movement of gas, and the combined gravitational and inertial effects in baffled dust chambers, as follows:

$$v = \frac{gd^2\rho}{k\eta} \qquad [1]$$

where v is the velocity of the particle with respect to the gas,
d is the diameter of the particle,
ρ is the density of the particle,
g is the acceleration due to gravity,
η is the viscosity of the gas,
k is the constant.

* See Fig. 8 on page 363 in paper by H. S. McKnight.

For inertial separation the formula is

$$v = \frac{d^2 \rho \gamma \omega^2}{k\eta} \qquad [2]$$

where ω is the angular velocity of the gas and suspended matter,
γ is the radius curvature of the path described by the gas,
k is a constant.

For baffled settling chambers where both gravitational and centrifugal forces are at work, these two formulas may be combined to yield the following equation:

$$D = k_2 \frac{gL}{V} \qquad [3]$$

where L is the length of the settling chamber,
V the gas velocity,
D the distance the particle moves out of the gas stream,
k_2 is a constant equal to $\frac{\rho d^2}{k\eta}$.

Electrical precipitators for a number of years were designed entirely from a set of empirical factors developed from experimental plant data.

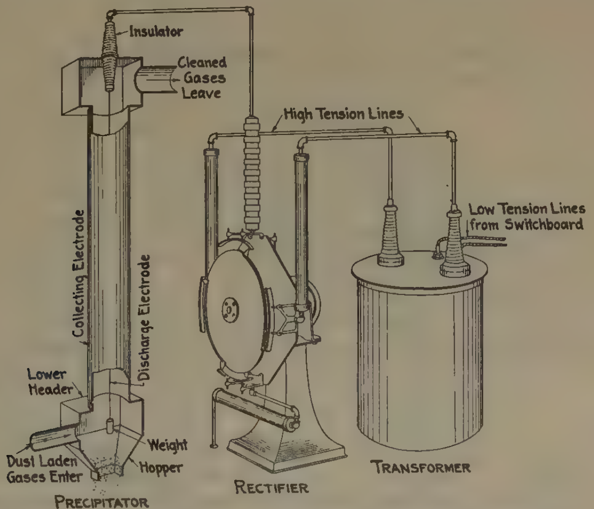


Fig. 9.—Diagram of Cottrell process and equipment.

Later Anderson(25,33,30) and Horne developed a formula of precipitator operation that permits all types of precipitators to be expressed in terms of one another and also permits the development of constants for design in any type of precipitator from the data accumulated by test work, pilot-plant work and actual plant operation. Likewise it provides a means of designing for any desired degree of plant efficiency.

The equation of precipitation(33) is given as follows:

$$1 - E = CK^t \qquad [4]$$

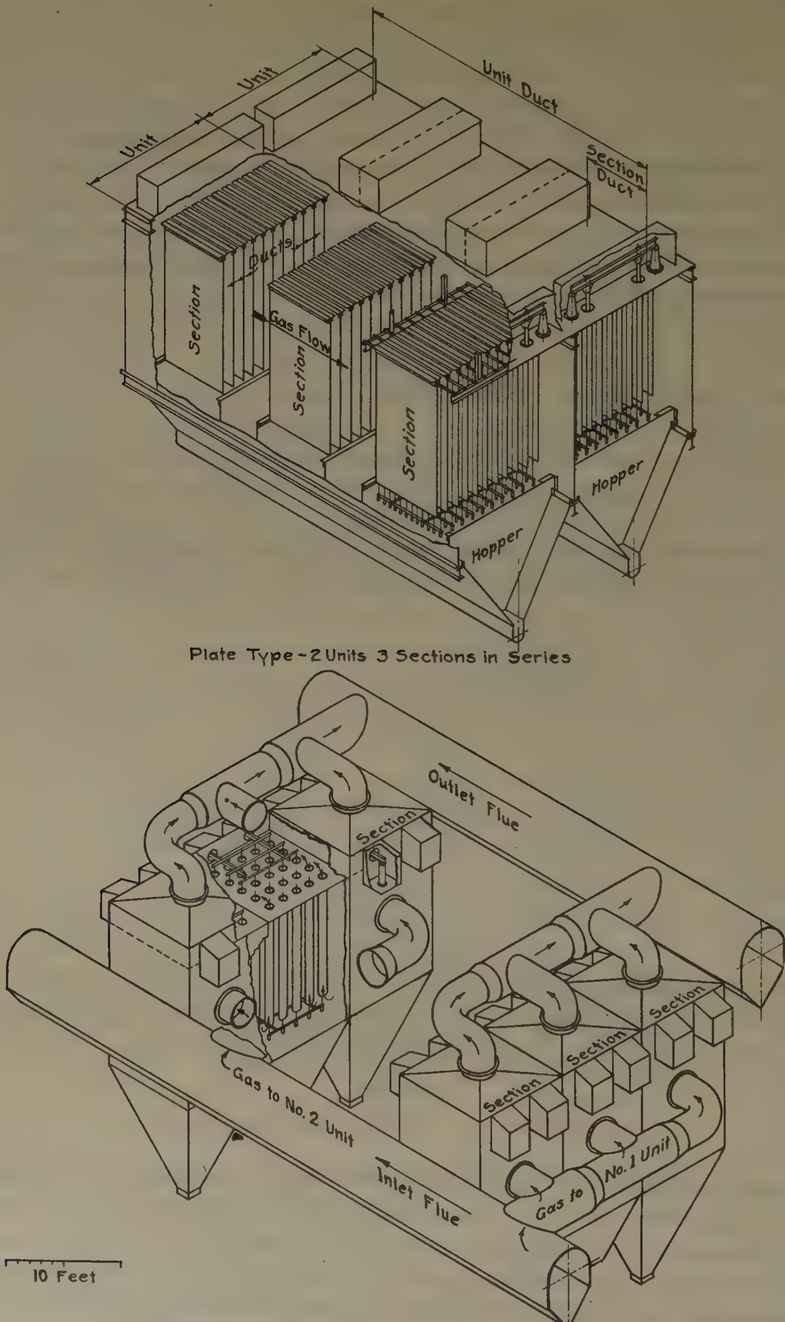


Plate Type ~ 2 Units 3 Sections in Series

Submerged Pipe Type ~ 3 Sections in Parallel

FIG. 10.—SKETCH ILLUSTRATING NOMENCLATURE IN GAS-TREATING PARTS OF AN ELECTRICAL PRECIPITATION INSTALLATION.

where E = precipitator efficiency,

t = time of treatment,

C = type of precipitator,

K = constant expressing the precipitation characteristics of gas, suspended and precipitated layer of fume or dust.

Precipitation efficiency is a logarithmic function of the time the gas is in the precipitator, and time of contact t may be expressed by the effective electrode length L divided by the gas velocity in the precipitator. The equation therefore, may be written

$$1 - E = CK^{L/V} \qquad [5]$$

Whence $L/V \log CK = \log (1 - E)$

From equation 5 it is evident the size of precipitator for a given set of gas conditions rises at an accelerating rate as the approach is made to 100 per cent, and doubles from 90 to 99 per cent, therefore it follows that for every installation there is an optimum size. A. A. Schmidt[25,27] has developed a formula for the most economical precipitation plant as follows:

$$x = a\left(\frac{\log \dfrac{b}{c \log d}}{\log d}\right) \qquad [6]$$

where a is the gas volume treated,

b is a function of the unit cost of precipitator, rate of interest and depreciation and cost of labor and power,

c represents the value of the solids carried by the gases,

d is a function of the specific precipitation rate for the fume considered,

x is the optimum size of the precipitator.

Regarding the equation of precipitation, W. A. Schmidt[33] makes the following comment:

> It should be emphasized that these equations pertaining to precipitation efficiency, apply only while normal precipitation is being performed. If a porous dielectric is accumulated upon the electrodes, and back ionization is established, normal precipitation is interfered with and these equations can no longer hold. It should also be emphasized that the precipitation constant is in itself a variable and that its value is only constant for a specific set of fume and gas conditions, and shifts with changes in gas composition, temperature, fume composition, fume concentration, physical state of sub-division of dust and fume, etc.

Horne[26] illustrates (Figs. 11 and 12) typical wiring connections for motor-generator and synchronous-motor electrical precipitation power sets. Potential between discharge and collecting electrodes which can be maintained with point or wire (a) negative and (b) positive, is illus-

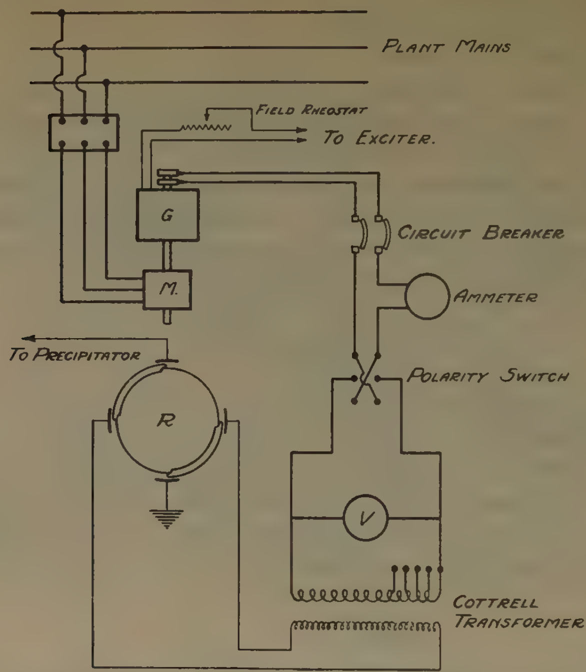


FIG. 11.—TYPICAL WIRING CONNECTIONS FOR A MOTOR-GENERATOR POWER SET.

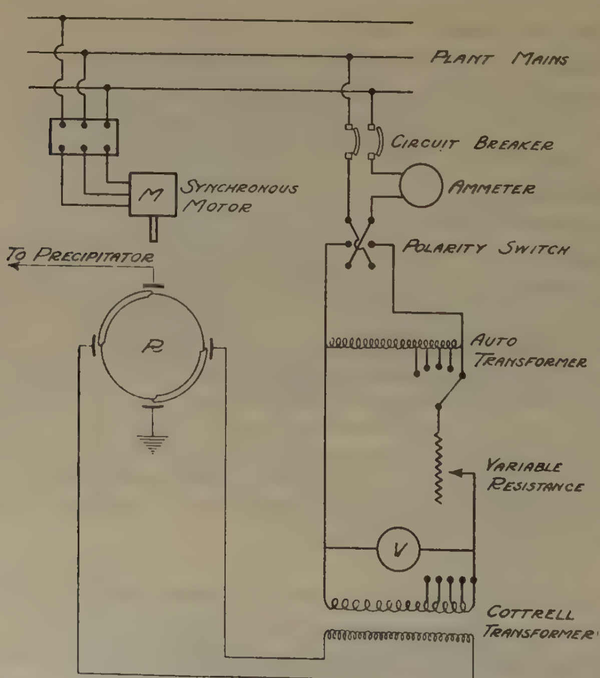


FIG. 12.—TYPICAL WIRING CONNECTIONS FOR A SYNCHRONOUS-MOTOR POWER SET.

trated by Fig. 13. Precipitation efficiency with voltage carrying up to the maximum is shown in Fig. 14.

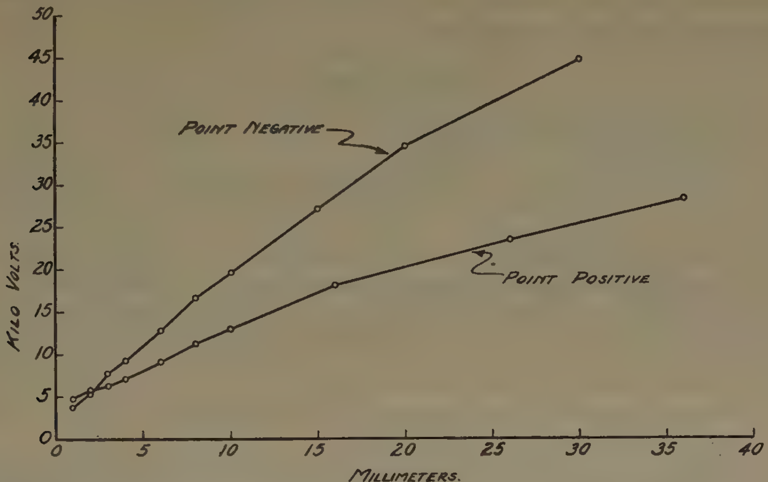


FIG. 13.—VARIATION IN MAXIMUM POTENTIAL MAINTAINED WITH DISCHARGE ELECTRODE: (a) NEGATIVE, (b) POSITIVE.

While American copper smelters up to the present have not entered the field of commercial recovery of the sulfur in their furnace charge as elemental sulfur, it is a problem upon which much work has been done and in Europe has now reached commercial operation in one or more plants.[41] The problem may be divided into two parts: (1) the recovery of the sulfur in its elemental form direct from furnace operations, that is, without passing through the formation of sulfur dioxide, and (2) the reduction of sulfur dioxide.

S. W. Young,[42] W. A. Hall[43] and others have worked on the reduction of sulfur dioxide by reducing oil gases, water gas, carbon monoxide and coke, sometimes with and sometimes without catalytic agents.

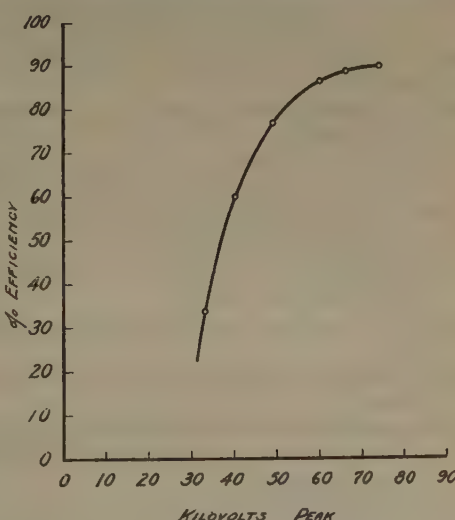


FIG. 14.—VARIATION IN PRECIPITATION EFFICIENCY WITH INCREASING VOLTAGES UP TO MAXIMUM.

While as a matter of scientific fact reduction of sulfur dioxide is not difficult, there is a good deal of difficult work ahead for our metallurgists before the problem is solved for a commercial operation, defining com-

mercial metallurgical operation as the recovery of products at less than the market value. It may be noted that elemental sulfur fume with proper humidity and temperature control is readily and satisfactorily collected by Cottrell equipment.

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Production of Arsenic Trioxide at Anaconda

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The recovery of arsenic was started at the Washoe Reduction Works (later called the Anaconda Reduction Works) in 1904, when an arsenic plant was constructed adjacent to the lower end of the 60-ft. main flue.[1] This plant originally consisted of a Brunton roasting furnace for treatment of flue dust and a small reverberatory furnace for treating the crude arsenic produced in the roasting operations, together with the necessary kitchens for condensation of the fumes and grinding and barreling apparatus for preparing the refined white arsenic for market.

The Brunton furnace was fired with wood. It was 14 ft. 6 in. in diameter with two outside fireboxes. The hearth revolved upon a central shaft. The charge was fed through a dust-tight hopper on to the center of the hearth and slowly moved to the outer edge of three rows of stationary rabbles fastened in the roof of the furnace. There was a 3-ft. space between the hearth and roof. A crude arsenic was condensed from this furnace.

The hot flame from the fireboxes passed directly over the charge, vaporizing the As_2O_3 that was carried through brick flues to the condensing chamber. This chamber was 8 ft. high, 16 ft. wide and 240 ft. long, divided into small kitchens by partitions spaced every 7 ft. It was connected with the main flue system to provide draft and to receive any losses from the chamber. The small kitchens were connected by openings in such a way that the fumes took a zigzag course, causing them to come in contact with the cold walls and ceiling and condense the arsenic. There was an opening from the outside into each kitchen, for the removal of the deposited arsenic, which was closed by a wooden door.

The refining furnace was a small coke-fired reverberatory having a hearth 10 by 20 ft. in dimensions. It had three hoppers through which the crude charge was introduced, and one firebox placed at one end. The hot gas from the burning coke passed over the charge, volatilizing the arsenic and carrying it to the kitchens.

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[1] A more detailed description of the early Anaconda arsenic plant may be found in an article by J. O. Elton: *Trans.* A.I.M.E. (1913) **46**.

The residue from both the roasting and refining furnaces was sent to the copper reverberatories for recovery of the copper, gold and silver contained therein.

During 1915 two furnaces similar to those now in use, fired with coke, were installed for both the roasting and refining operations.

In 1917, it was decided to construct Cottrell treaters and a reverberatory furnace in connection with the building of the present stack. The treaters consisted of four units of five treaters each, the dust so collected to be treated in a reverberatory furnace. The gases from the reverberatory were treated in a special hot treater to remove most of the dust, then in a cold treater to deposit a high-grade crude arsenic before the gases were discharged through the stack. The crude from these treaters ran about 75 per cent arsenic and was treated direct in the refining furnaces for the production of refined white arsenic. This method was used until 1922, when this 75 per cent crude was taken to roasters where a high-grade crude was produced for the refining plant. These roasters were McDougall copper furnaces altered as described later in this paper.

In 1924, water sprays were being used to cool the reverberatory gases before they entered the treaters. Of 500 tons of water used per day, 400 tons was vaporized, while the remaining 100 tons, heavily laden with dust and arsenic, seeped from the bottom of the flue and was settled in ponds. This gave a good settlement and a clear overflow, but the overflow contained about 3½ tons of arsenic. Several schemes were tried to recover this arsenic, but the most feasible scheme, and the one used, was to filter the entire amount in a Shriver filter press, send the cake to the arsenic roasters and use the filtrate as part of the original spray water.

In July, 1927, with the introduction of selective flotation at the concentrator, the stack reverberatory was abandoned.

In 1931, when gas was brought to Anaconda, the firing of the roasters was changed from pulverized coal to gas, and of the refining furnaces from coke to gas, which has proved to be an ideal fuel for both of these operations. Since that time, the operations have been as outlined later in this paper.

The Present Plant

The present arsenic plant, aside from the main flue system and Cottrell treaters at the stack, consists of the following:

In the old McDougall roasting building, four 16-ft., six-hearth furnaces have been remodeled. Each one is equipped with a dust-tight, two-compartment feed hopper. A Link-belt pan conveyor entirely enclosed under this hopper conveys the feed to the furnace. Individual gates on each compartment permit almost any proportion of the different

feeds desired. A second hopper permits feeding coal or flotation concentrates to aid in the reduction of arsenates. The furnaces are fired with gas directly through the shell on the third and fifth hearths. The residue discharges from the outside of the sixth hearth to hoppers and thence to cars for delivery to the reverberatory department, or to be stock-piled and later sold for its lead and bismuth contents.

Each of these furnaces discharges its gases, laden with arsenic and some dust, to three condensers operated in series. These condensers are adjacent furnaces from which the intermediate hearths have been removed and in which baffles have been placed down the centers, leaving the roofs and sides intact and the arms and rakes operating on the sixth hearth. The last condenser in each series is connected through a cross flue to the McDougall dust chamber and thence to the main flue.

The refinery consists of two gas-fired reverberatories (12 by 34 ft.) so arranged that either one can discharge its gases to any one of the three rows of kitchens.

The crude arsenic is delivered from the roasters in cars and these are discharged into a small bin which in turn feeds an elevator that discharges to a screw conveyor running lengthwise of the furnace. Five hoppers down the center of the furnace discharge the crude arsenic to the hearth of the furnace.

The grinding and barreling plant consists of a small bin directly over a screw conveyor to which the refined arsenic is delivered from the kitchens by wheelbarrows. The screw conveyor delivers to an elevator that discharges to a Hum-mer screen, the undersize going directly either to barrels or to the Dracco apparatus for bulk shipment. The oversize goes to a Sturtevant vertical Rock-Emery mill and from there to a screw conveyor that delivers to the original elevator, thereby forming a closed circuit.

All points at the refinery where there might be leakage of dust or fume are enclosed and are connected to an exhaust fan that discharges into the main flue.

The barrels are filled by a bumping apparatus consisting of a cylinder and necessary valves operated by compressed air. A leather connection from the bin makes an airtight connection to the barrel.

In the lower portion of the bin just above the leather boot is a 4-in. pipe slotted on top and leading to the Dracco apparatus. This arrangement makes it very simple to change from barrel to bulk loading. All that is necessary is to start either one and leave the other inoperative.

The Dracco apparatus for bulk loading is standard apparatus, except that instead of the small collecting tank with rotary lock there is a large tank of 35 tons capacity with a plug valve at the bottom. This tank is elevated high enough over a railroad track so as to permit the loading of the largest tank car. Air jets in the cone of the tank, to loosen the

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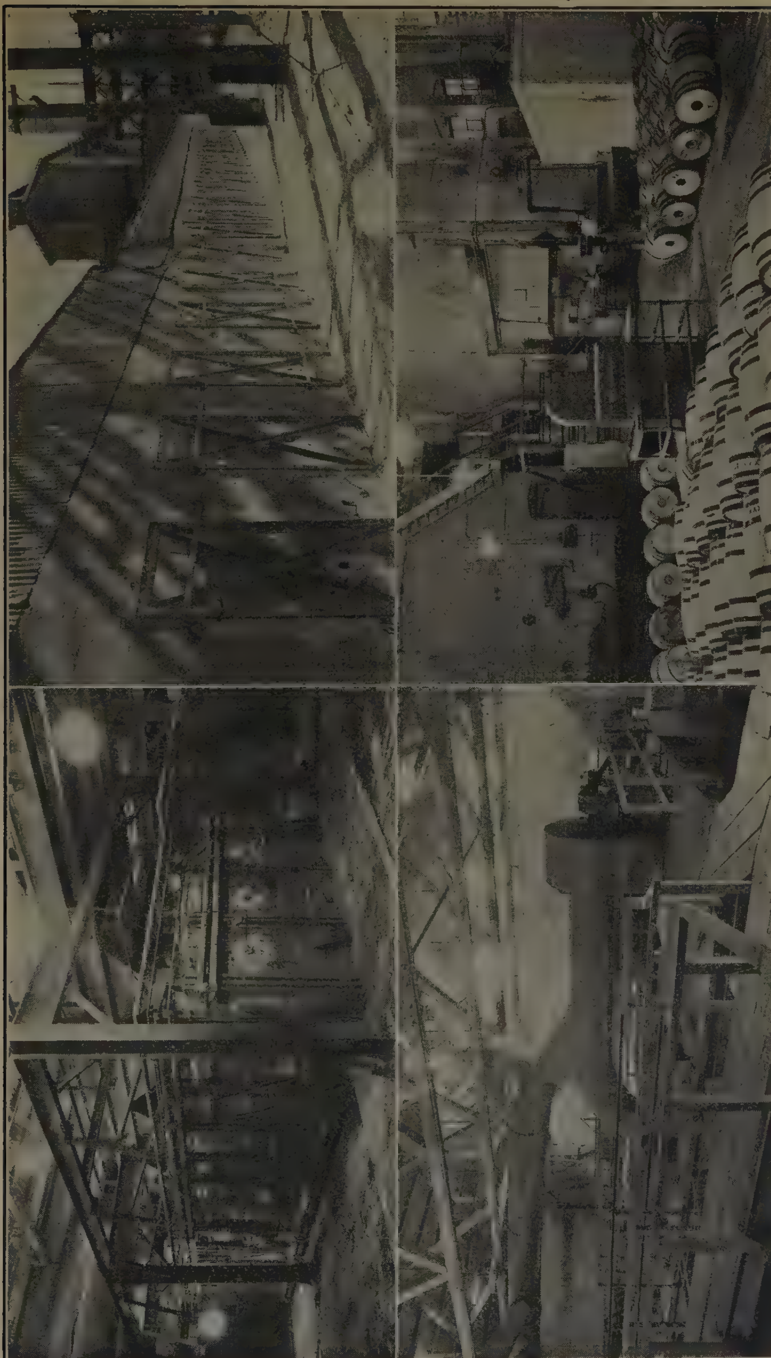

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FIG. 1.—GAS-FIRED REVERBERATORY REFINING FURNACE, ARSENIC PLANT, ANACONDA REDUCTION WORKS.
FIG. 2.—VENTILATION SYSTEM OVER REFINING FURNACES.
FIG. 3.—WEST ROW OF ARSENIC KITCHENS.
FIG. 4.—BARRELING ROOM. GRINDING AND SCREENING MACHINERY IS BEHIND PARTITION AT RIGHT.

contents, make it possible to drop the entire load into a tank car in 10 minutes.

The Process

Dust from the main flue is put into two large storage bins and from there trammed to the roaster plant. The resultant crude is usually not of high enough grade for the refinery and is held either in cars or bins until enough has accumulated for a re-run campaign. The low-grade crude is mixed with main flue dust (usually from the 120-ft. flue), using the two-compartment hopper in such proportions as to give a feed running 55 to 60 per cent As_2O_3. The crude from this run usually contains 93 to 95 per cent As_2O_3, and is suitable for the refinery.

Dust from the Cottrell treaters at the stack is delivered directly to the roasters in standard bottom-dump railroad cars. This dust is so light that a 50-ton car holds only 15 tons of dust. It constitutes the main arsenic supply, and is the best feed to the arsenic roasters. Mixed with coal or flotation concentrate for reduction, it gives a residue seldom containing over 5 per cent arsenic and a crude assaying 95 to 99 per cent arsenic.

The refining operation is a batch process. Starting with a clean hot furnace, the floor is first covered with a thin layer of sand. This prevents the sticking of the residue to the hearth. Then 1½ to 2 tons of high-grade crude is charged and leveled off, the furnace is closed and the gas burners regulated to maintain 950° F. in the gases as they leave the furnace. The charge is cooked until, by inspection, the arsenic is mostly volatilized. In a slow-cooking charge, the arsenic on the hearth may be lifted and broken up to expose new surfaces. When finished, another charge of 1½ to 2 tons is dropped and the operation repeated. Most of the charge is dropped from the first three hoppers; about three charges a day from the fourth and only one per day from the fifth.

At the end of 24 hr. the residue remaining on the hearth is worked to the end of the furnace, where it drops into cars to be taken to the copper reverberatories.

Usually only one refining furnace is in operation at one time and will produce 20 tons of refined arsenic per day. One row of kitchens will hold about 200 tons and is emptied by wheeling the arsenic to the grinding apparatus where 25 tons in 8 hr. is the usual capacity, whether to barrels or in bulk.

The Anode Department of the Noranda Smelter*

By W. B. Boggs† and J. N. Anderson,‡ Noranda, Quebec

(New York Meeting, February, 1934)

Originally, the copper produced at the Noranda smelter was shipped in the form of blister bars to the Laurel Hill refinery of the Nichols Copper Co., New York. In 1930 a refinery was built at Montreal East to refine Noranda copper and copper from the Hudson Bay Mining & Smelting Co., at Flin Flon, Manitoba.

After investigation of the probable costs of making anodes at Noranda and at Montreal, it was decided to make them at Noranda rather than to continue to make blister bars there. Fuel is a major item of expense in making anodes, and since less fuel would be required in making them from molten copper than from blister bars, a lower anode-furnace cost could be expected at Noranda than at Montreal. Further, the expense connected with making blister bars at Noranda would be eliminated. The cost of treating anode and wire-bar furnace slags is a serious item in a refinery, and it was found that the cheapest method would be to ship such slags back to Noranda. With the anode furnace at Noranda, much less anode-furnace slag would be made at the refinery.

Against these savings effected by making anodes at Noranda must be set the high operating cost of an anode furnace at Montreal, using only tank-room scrap and the Flin Flon copper. This furnace would be handling a comparatively small tonnage and would have a high cost per ton.

The usual equipment for making anodes from liquid copper consists of either a stationary reverberatory furnace or a tilting Tacoma-type furnace together with an anode wheel of the Clark or Walker type. The reverberatory type of furnace has been developed from the furnace used in the old Welsh process of smelting and refining copper, while the tilting furnace is a development of the holding furnace, which is essentially a Peirce-Smith converter without tuyeres and provided with burners.

Refining as carried out in either of these furnaces is a batch process; that is, several converter charges are accumulated and then refined and cast. Colonel H. H. Stout recently developed in Arizona a so-called

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continuous process of refining. The equipment consists of three cylindrical vessels lined with a basic refractory and well insulated, each vessel being of sufficient size to hold one converter charge of copper. Two of the vessels are stationary, while the third runs on rails along the length of a number of stationary molds. Refining is done in the two stationary vessels, which are heated by oil burners, alternate vessels being used for successive converter charges. When a charge has been refined, it is transferred into the third or casting vessel, from which the copper is poured into the anode molds. The refining and casting vessels are tilted by means of hydraulic rams. Enough molds are provided to hold all the copper from one charge. The advantages claimed for this method are savings in fuel, poles and labor, as well as better casting results due to the use of stationary molds.

In the early part of 1930, J. R. Bradfield, plant engineer, and one of the writers visited several smelters and refineries in the United States, and made a study of the three methods of making anodes: (1) using a reverberatory furnace with anode wheel; (2) using a Tacoma-type furnace with anode wheel; (3) the Stout process. The result was a decision to install a reverberatory furnace with a Walker wheel. The Tacoma-type furnace did not appear to have any great advantage over the reverberatory furnace and had the disadvantage that it is difficult to work charges of impure copper.

In order to keep down the capital cost, it was desired to put the anode equipment in the space originally occupied by one of the blister-casting machines and part of the shipping aisle. It was found that this would be possible with a reverberatory furnace and wheel, but that the continuous system, with its three vessels and a long row of stationary molds, would require expensive alterations and extension to the existing buildings. Further, since the converters in use at Noranda are large Peirce-Smith converters, the vessels required would be very large and expensive. In addition, this method did not seem to be suited to handling copper with many impurities to be eliminated. Noranda copper is very low in impurities, but it was felt that future possibilities must be taken care of, either with depth development in the mine or from custom material. The low operating costs that might be expected with the continuous system did not seem to justify the higher capital cost of the equipment and the other disadvantages.

Fig. 1 shows the general arrangement of the anode department with its relation to the rest of the smelter. Molten blister copper is transferred from the converters to the anode furnace by the same cranes that serve the converters. There is one anode furnace designed to hold 150 tons. At present, because of alterations, the capacity is 120 tons. Space was provided for a second similar furnace should copper production be increased.

The refined copper is tapped from the furnace and runs through a launder into a small ladle, from which it is poured into the molds on the casting machine or anode wheel. The anodes are cast one at a time, the wheel being brought to rest with a mold under the ladle, and the ladle tilted by an air cylinder. As the wheel rotates, the anodes are cooled by water sprays and then lifted off into a bosh. From the bosh, the anodes are taken to the shipping platform where they are inspected, trimmed and loaded into box cars. A 10-ton overhead crane operates the full length of the shipping aisle and is used for transferring anodes from the bosh to the platform, for handling molds and for bringing poles and other supplies to the anode furnace.

The design and the construction of the anode department were directed by J. R. Bradfield. No essential changes have been made to the original design, except for the furnace bottom and side walls, where changes were necessitated by the failure of the silica-brick bottom.

Anode Furnace and Casting Wheel

The anode furnace is built on a concrete base supported on piers, so that there is an air space underneath. The side walls up as far as the metal line are reinforced by heavy cast-iron plates. The buckstays are set at intervals of from 3 to 4 ft.; tierods are provided at the top and bottom of the buckstays and are kept at a constant tension by springs; the bottom tierods run through pipes set in the concrete base. Springs on the tierods of an anode furnace are a great advantage, as they do away with the necessity of adjusting the tierods when heating or cooling the furnace.

In the original construction the top of the concrete base was made concave and the brick bottom built as an inverted arch. Two layers of fireclay brick, 4½ in. and 9 in. deep, respectively, were laid first; on top of this was spread a 1-in. layer of silica sand. On this was laid a course of 9-in. silica brick and then a course of 15-in. silica brick, these courses being laid as inverted arches. The method of tying in the bottom to the side walls is similar to the practice followed in the Michigan copper furnaces. The side walls to a point 3 in. above the metal line were built of magnesite brick, 19½ in. thick and faced on the outside with 4½ in. of silica brick. From this point up to the skewbacks, the side walls were of silica brick, 21 in. thick. The arch was of 15-in. silica brick throughout, except at the verb, where chrome brick was used. The uptake was built of fireclay brick. The bridge wall was built of magnesite brick and the firing wall of fireclay brick.

As shown in Fig. 2, the original construction of the furnace has undergone several changes. A basic monolithic bottom, a description of which will be found later in the article, has replaced the former silica

bottom. The arch of the furnace at present is constructed of 20-in. silica brick. The flue is built of silica and the verb of magnesite brick. The side walls are now constructed of unburned magnesite brick faced on the outside with silica brick.

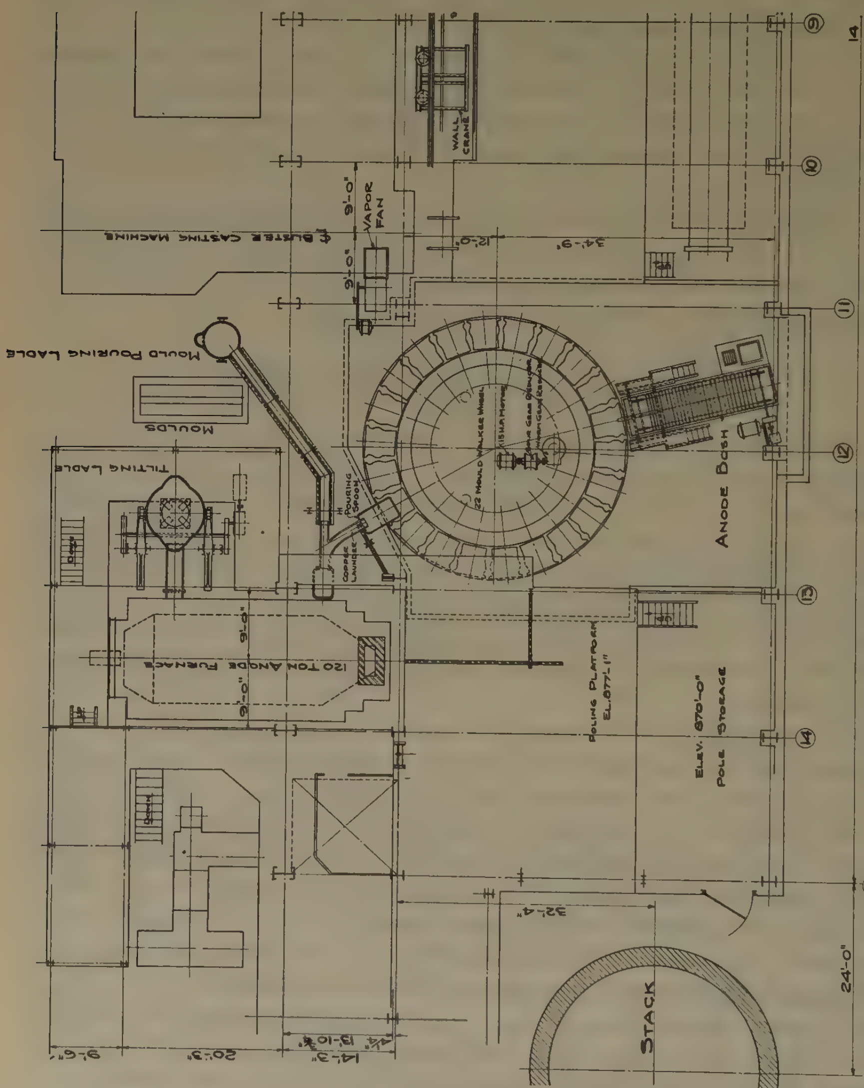


FIG. 1.—GENERAL ARRANGEMENT OF ANODE DEPARTMENT, NORANDA SMELTER.

The inside dimensions of the furnace are 28 ft. 3 in. from bridge wall to the front and 11 ft. across at the widest point. The arch is comparatively low, since only liquid copper is charged into the furnace.

The furnace tapping slot is close to the front end of the furnace and the bottom is sloped so that the lowest point is at the slot. The drop in

the launder from the tapping slot to the end is 12 in. and the end of the launder is 2 ft. 11 in. above the top of the molds on the wheel. A brick bridge is provided at the end of the launder so that charcoal can be kept

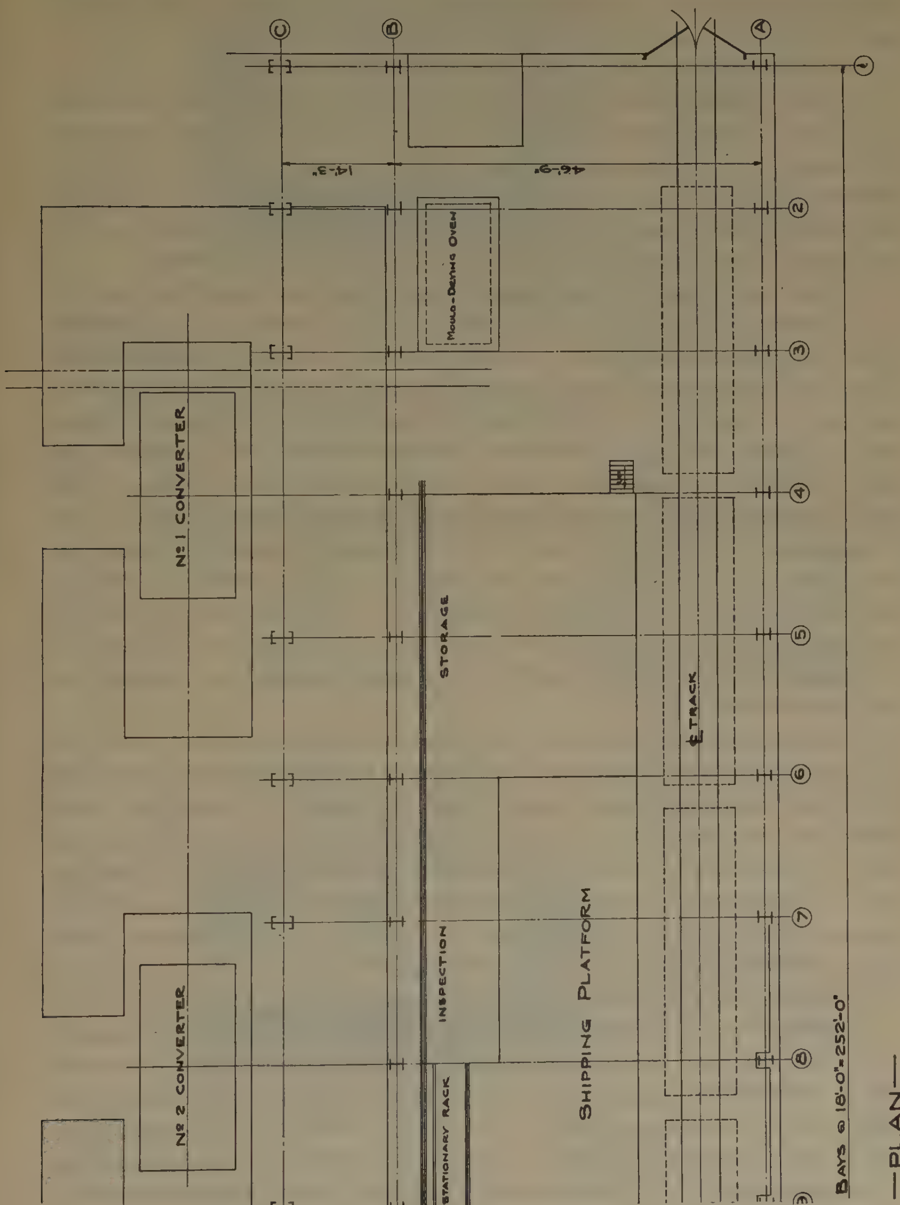


FIG. 1.—(*Continued*).

on the stream of copper. The pouring spoon or ladle will hold about 1000 lb. of molten copper. It is operated by a balanced air hoist, the

cylinder of which is 5 in. in diameter with a 48-in. stroke. When the ladle is in the pouring position, the top edge of the lip is 6 in. above the casting surface of the mold. The lip of the ladle is 30 in. wide and the mold is 36 in. wide inside. The stream of copper strikes the mold 12 in. from its lug end. The bridge in the ladle is made of firebrick keyed in place and is 1½ in. above the bottom of the ladle and 14 in. from the lip.

In most anode plants, the furnace is on the floor level and the wheel is in a pit. Although this is convenient, it has serious disadvantages; it is difficult to drain the pit and, should the copper break out of the furnace and come in contact with the water, there would be a dangerous explosion. The furnace at Noranda was set well above the floor level and the floor was provided with plenty of slope to give drainage.

The casting wheel is of the Walker type, with several modifications and improvements suggested by an inspection of the various types of wheels in use at anode plants in the United States. To obtain good casting results, it is essential to have a positive method of drive, smooth acceleration and no backlash. In the wheel as installed, a 15-hp. motor drives a vertical pinion through a horizontal and a vertical speed reducer, the pinion engaging a gear on the inside of the main wheel casting. Two guide pinions were provided, fitted with brakes operated by an air cylinder and controlled by a pedal at the operator's platform, but it has been found that the brakes are not necessary. This method of driving the wheel has proved entirely satisfactory. The wheel has an outside diameter of 33 ft. and weighs, without the molds, 69 tons. There are 22 molds, weighing 6300 lb. each, so that the total weight of the machine and molds is 138 tons.

When anodes are being cast, the copper runs continuously from the furnace into the pouring ladle. The wheel is brought to rest with a mold under the ladle and copper poured into the mold by tilting the ladle. When the mold has been filled to the proper depth, the ladle is tilted back and the wheel rotated so that the next mold is brought under the ladle. The anodes are cooled by water sprays set over the molds, beginning at a point four molds from the pouring ladle. The spray pipes are set tangentially to the circumference of the wheel, and a vapor hood connected to a suction fan is set over the sprays.

Preparatory to transferrence of the anode from the wheel to the bosh, the lug end is raised about 8 in. from the mold by a special lifting mechanism, set about two-fifths of the circumference of the wheel from the pouring ladle. The requirements for this device are that it be entirely automatic in operation, positive in action, and that it must not distort the anodes even when they have a tendency to stick to the molds.

The design of the device used was adapted from a similar one in use at another plant. Each mold is cast with two countersunk holes in which

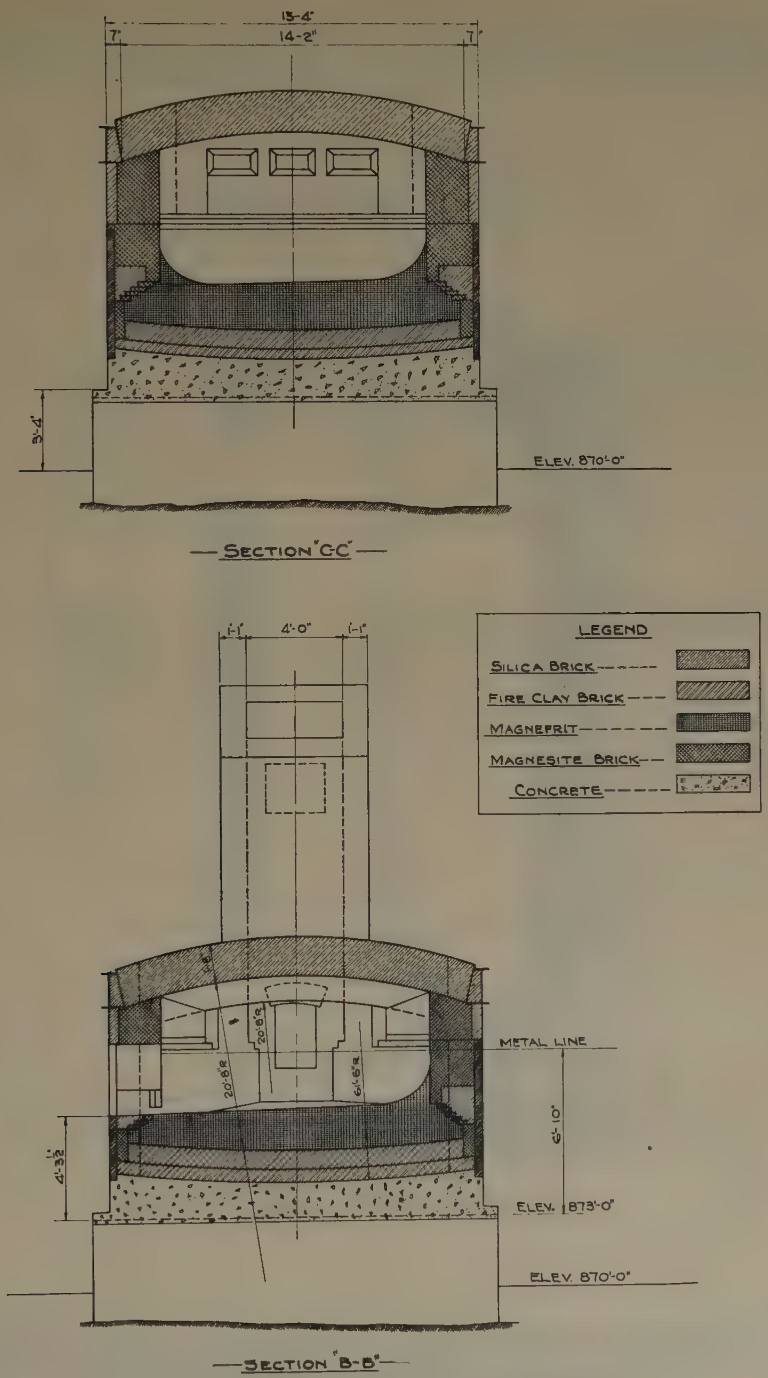
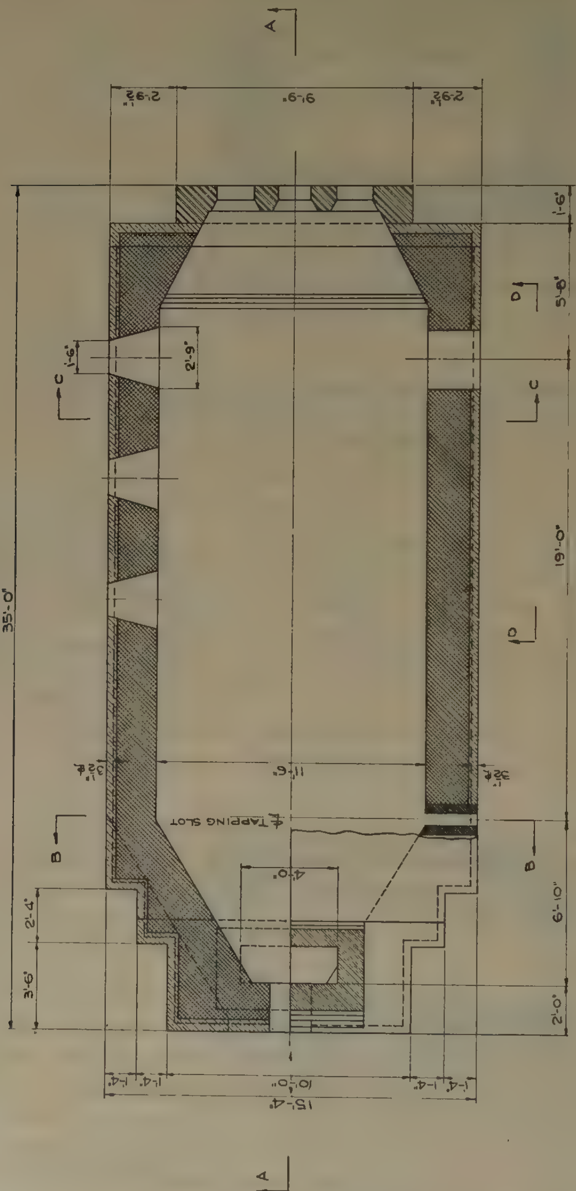


FIG. 2.—PRESENT ANODE-FURNACE BRICKWORK.

are placed loosely fitting cast-iron pins with tapered heads, which fit the countersunk holes and rest flush with the casting surface. The lower ends of the pins project about 6 in. through the bottom of the mold. A

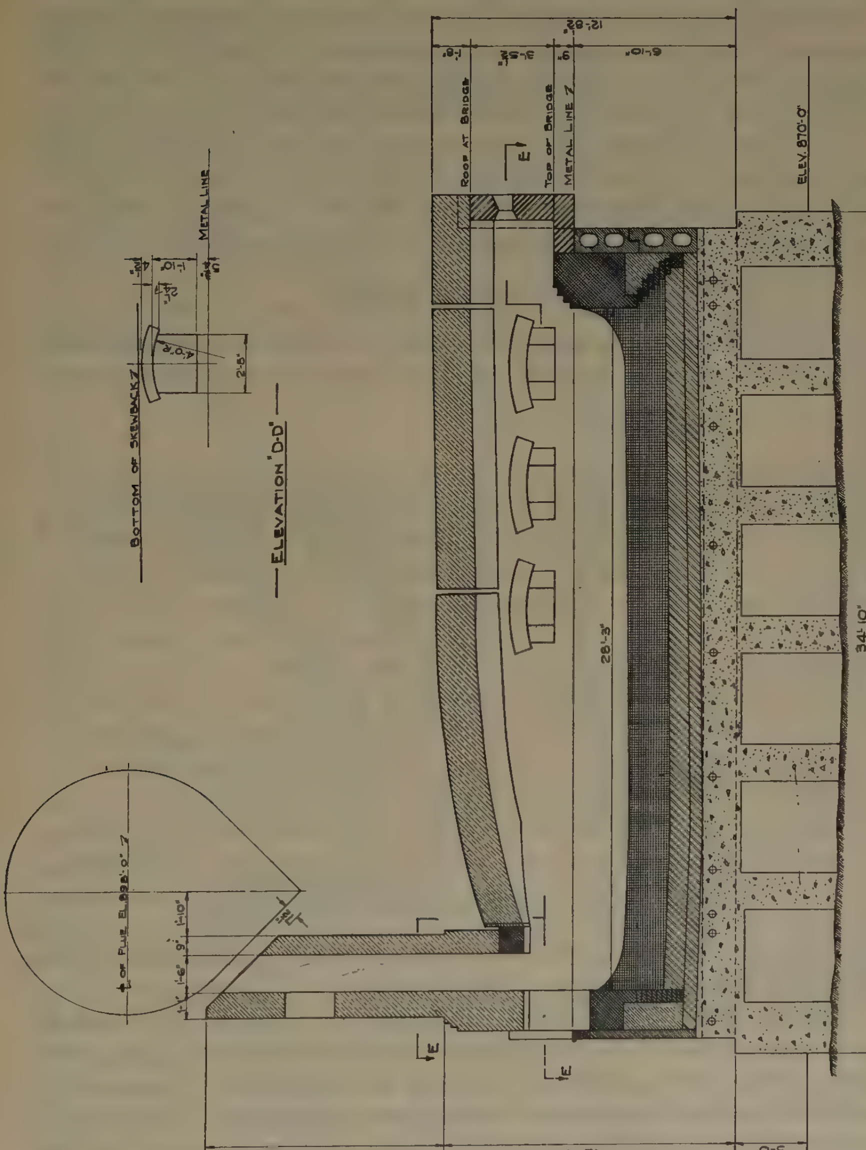


—PLAN SECTION E-E—

FIG. 2.—*(Continued).*

double-acting air cylinder, 10 in. in diameter with an 8½-in. stroke, is placed below the wheel, two-fifths of the circumference of the wheel from the pouring ladle. The top of the cylinder piston is an I-beam and

the position of the cylinder is such that when the wheel is stopped to pour an anode, the two pins in a loaded mold are directly over the I-beam of the piston. The up and down strokes of the piston are controlled by a



SECTION 'A-A'

FIG. 2.—(*Continued*).

two-way valve connected with the air cylinder and operated by a series of shoes, one for each mold, which are attached to the wheel frame. These shoes are so placed that as the wheel is stopped to cast an anode a

shoe is actuating the two-way valve so as to raise the piston, which is then directly below the two pins. The piston forces the pins up, thus raising the lug end of the anode until it is clear of the mold. The length of the shoes is such that when the wheel again starts to move the valve is held open until both pins slide from the I-beam on the piston on to a supporting track for the pins, thus maintaining the anode in its tilted position so that it can be lifted off as it passes the bosh. When the shoe has passed the valve lever, a counterweight reverses the valve and the piston moves downward and is in position for another stroke. The supporting track for the pins continues some distance past the bosh, and when the pins pass the end of the track, they fall by gravity into their normal position, so that the mold is ready for the pouring of another anode.

Transferring Anodes

As the molds pass the bosh, the anodes are lifted off by an air hoist, the position of which is shown in Fig. 1. The hoist is hung from a crawl

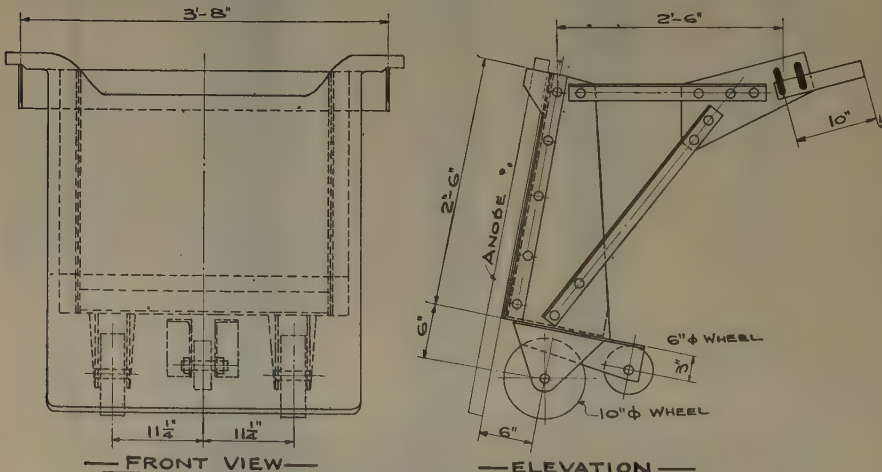


Fig. 3.—Details of hand truck for moving anodes.

running on an overhead I-beam and is provided with a special hook that fits under the anode lugs.

Running water flows through the bosh, which has a capacity of 35 anodes, carried at 4½-in. intervals by chains on either side of the tank. These chains are supported on a track and are driven by sprockets connected to a motor through suitable reduction gearing. The control of the motor is near the hoist, so that it can be operated by one of the two hoistmen. The anodes are transferred from the bosh to a rack in the shipping platform by an overhead crane using a hook that lifts six anodes at a time. The top of the rack is at the shipping-floor level. It holds 83 anodes.

The standard weight of an anode is 710 lb. and the refinery allows 5 per cent over or under this weight. Each anode must be handled separately in order to check its weight and to allow for inspection and trimming. A wall crane was installed to do this, and to transfer the anodes from the rack to hand trucks on which they could be loaded into box cars. Details of the hand trucks are shown in Fig. 3.

Two electric lights are provided, with contacts on the lifting rack of the crane. These contacts are adjusted so that if the anode is more than 5 per cent underweight, both lights burn; if the anode is more than 5 per cent overweight, neither light burns; if the anode is within the weight limits, one light burns. Light and heavy anodes and anodes with bad lugs are piled at one end of the shipping platform, to be taken to the converters and remelted. The shipping platform is at the same level as the floor of the box cars, and after inspection and trimming the anodes are loaded into the cars.

Molds

Cast-iron molds were used at the start, as no refined copper was available and blister copper is unsuitable for making anode molds. After the furnace had been started, molds were made from anode copper, using a cast-iron master mold. These molds were far from satisfactory because the heat and weight of the molten copper warped the core plate and lugs of the master mold, and the resulting molds were out of shape and did not wear well. Molds are now cast in sand, a method of making copper molds recently developed by a refinery in the United States, from which Noranda practice was copied.

A wooden pattern is used, shown in Fig. 4. This pattern is set on a cast-iron baseplate, to which side plates are then bolted on. Moist molder's sand is carefully packed around the pattern, using an air-operated tamping hammer; a top plate is then bolted on, the mold is turned upside down, the baseplate removed, and the wooden pattern taken out. The mold is then dried in a coke-heated oven for 24 hr. and is ready for the copper to be poured into it. When molds are to be made, the furnace charge is poled until the oxygen content is about 0.04 per cent.

The molds are 12 in. thick and weigh 6300 lb. each. They are set on the machine so that there is a slope of ⅛ in. toward the lug end of the mold. This is done to secure rapid flow of copper into the lugs, which is necessary to make anodes without defective "Baltimore groove" lugs.

Each mold will last, on the average, for 30 charges, which represents 500 anodes, or 170 tons per mold. Individual molds have lasted for as many as 60 charges. In service, the molds develop cracks where the stream of copper strikes the surface, and sag slightly in the center.

Furnace Firing and Fuel

The furnace is fired by pulverized coal supplied by a Kennedy Van Saun unit pulverizer situated in the reverberatory furnace coal plant

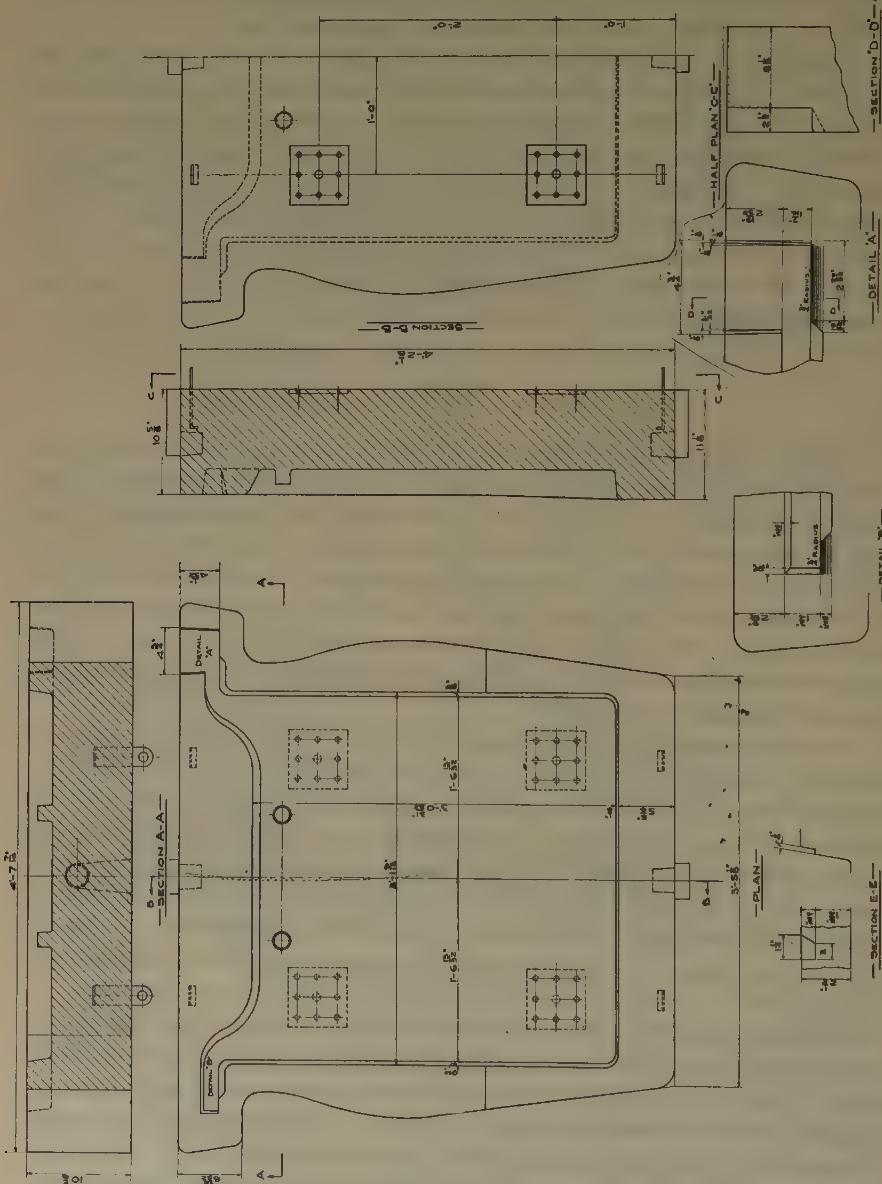


Fig. 4.—Wooden pattern for mold.

about 500 ft. from the anode furnace. Several alternatives were considered before this installation was decided on. One was a small pul-

verizer with a storage bin either at the coal plant or at the furnace. It was decided, however, that the advantages of a storage system did not justify its higher first cost. Another alternative was a unit pulverizer at the anode furnace; this, however, would have necessitated providing for the unloading and storage of raw coal, and also an operator. With the pulverizer in the coal plant, no additional unloading or storage facilities were required and the unit could be operated by the operators who care for the reverberatory furnace pulverizers.

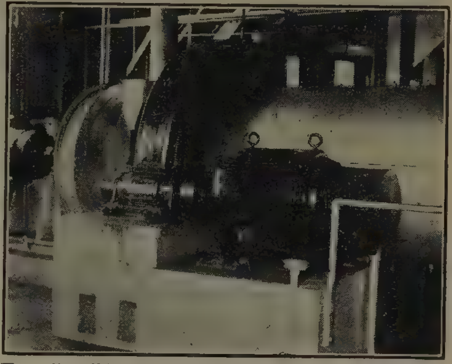


Fig. 5.—Kennedy Van Saun unit pulverizer.

A photograph of the unit pulverizer is shown in Fig. 5. An 80-ton storage hopper was provided, the bottom having two discharge points so that a second pulverizing unit can be installed. The coal is fed into the ball mill by a plate-type feeder driven by a variable-speed direct-current motor, the speed range being about 4 to 1. The ball mill is 8 ft. long and 3 ft. 6 in. in diameter, and carries a load of 5800 lb. of 2-in. cast-steel balls. It is driven by a 25-hp. slip-ring a.c. motor, which is wound for speeds from 750 to 1500 r.p.m. Air for conveying the coal passes through a steam heater and enters the ball mill at the opposite end of the fan.

The fan is a No. 55-C Simplex and consists of a boilerplate casting with cast-iron liners. It is driven by a 15-hp. d.c. variable-speed motor with a speed range of 900 to 1800 r.p.m. In calculating the volume and pressure requirements of the fan, it was estimated that, in order to convey pulverized coal over such a long distance, 60 cu. ft. of air would be required per pound of coal and that the minimum velocity in the pipe should be 53 ft. per second. Tests have shown that these figures are conservative and that 40 cu. ft. per pound of coal and 40 ft. per second are sufficient.

The coal used is Pennsylvania low-sulfur coal of the following analysis: volatile matter, 29.38 per cent; fixed carbon, 63.83; ash, 6.79; sulfur, 1.22; B.t.u. per pound, 14,527. It is a screened coal, being all minus ⅜-in. mesh. Coal is received in gondola-type cars and goes directly to storage. It is dried as required and conveyed to the storage bin over the ball mill.

Two pipes, 5 and 8-in. diameter, respectively, were provided for conveying the pulverized coal to the anode furnace, the smaller to be used for low coal consumption and the larger for high coal consumption. The fan discharges into a 9-in. pipe and, by means of a Y and a deflector plate, the coal and air stream can be sent into either the 5-in. or 8-in.

line. The furnace has three fantail burners, the two outside ones being fed by the 8-in. line and the middle one by the 5-in. line. It has been found, however, for ordinary operation that the single center burner is sufficient. The two outside burners are rarely used, except to obtain high temperatures when fusing a patch in the bottom. Secondary air for the combustion of the coal is introduced around the burner by a fan that blows the air into a rectangular metal wind box that covers the firing end of the furnace. The amount of secondary air is controlled by opening or closing ports in the wind box.

Present Operation

After a charge has been cast, the skimming bay is cleaned of slag and copper accretions and is then coated with a mixture of 90 per cent Quebec magnesite cement and 10 per cent fireclay mixed with water to make a smooth mud. The tapping slot is thoroughly cleaned and then filled with a mixture of two parts of screened sand, two parts of fine coal and one part of fireclay. This material is well mixed beforehand and water is added until the mixture is fairly "stiff." The tapping slot is filled with this mixture and the tapping bars are put in place. Several hundred pounds of sand are put into the furnace under the end of the launder where the liquid copper enters, to serve the double function of protecting the bottom and fluxing the copper and impurities which are oxidized in the refining of the charge.

Copper production at present varies between 90 and 100 tons a day. As each furnace charge is approximately 120 tons, this means casting six times a week. The copper is blown to high blister in the converters, each converter charge containing from 60 to 70 tons of copper. Converter operation is generally so arranged that, on the days when copper is to be cast, the furnace is filled between six and ten o'clock in the morning. The height of the skimming bay can be adjusted, by changing the number of bars, for a range in the size of the charge from 90 to 120 tons.

In fire-refining of blister copper containing small amounts of impurities, air is blown into the bath of molten copper until it is saturated with oxygen, with the object of oxidizing the impurities. Sulfur is driven off as sulfur dioxide, and metallic impurities, together with some of the copper, are oxidized and float as a scum on the surface of the bath. Silica sand is added and unites with these metallic oxides to form a viscous slag, which may be readily skimmed.

Noranda blister copper is comparatively low in impurities and no special treatment is required beyond that outlined above. Three ports are provided on one side of the furnace for the air pipes, and as soon as the furnace is filled the air pipes are put in and skimming of the charge begins. The slag is skimmed into small pans resting on trucks. As soon as a pan has been filled it is drawn to the converter aisle and charged

into one of the converters. About 2 tons of slag is made from the average charge of 120 tons of copper. Skimming takes about an hour and then the charge is "flapped," or splashed with a rabble, through the skimming door; this is an effective method of decreasing the time for saturating the bath with oxygen. Blowing and flapping are continued until a well defined crack is seen on several successive button samples taken with a small "say" ladle, which indicates that the copper is saturated with oxygen. In this condition it is known as "set" copper, and contains about 0.9 per cent oxygen.

As soon as the copper is at "set," the pipes are taken out, the flapping rabble removed and the charge skimmed clean. Formerly about 500 lb. of low-sulfur coke was shoveled on to the bath to prevent further oxidation of the copper. Recently, because of the difficulty of obtaining low-sulfur coke, local cordwood has been substituted advantageously. The copper is then poled to deoxidize it ready for casting. Birch poles, 25 ft. long and ranging in size from 6 to 10 in. at the small end, are used. Seven such poles, as an average, are required to deoxidize a charge from 0.9 per cent oxygen to 0.05 per cent oxygen. The oxygen content of the bath is judged by the appearance of the button samples taken with the "say" ladle. As it decreases from that of set copper, the surface of the sample becomes flatter and a black speck, called a "nigger," forms in the center of the surface as the sample solidifies.

The "set" or oxygen content of the copper can be closely estimated by the appearance of the surface of the sample and the size of the "nigger." At 0.08 per cent oxygen, the surface of the sample is almost flat and the "nigger" has almost entirely disappeared. At 0.05 per cent oxygen, the "nigger" has entirely disappeared and the surface of the sample is flat.

The oxygen content of the anode copper was formerly determined by examining a polished etched sample microscopically. Since the operators have become more experienced, the daily determinations have been discontinued.

When the charge has been poled to the correct oxygen content, it is cast into anodes. The average casting rate is 93 anodes per hour, equivalent to 32 tons of copper per hour. The anodes are cast with the "Baltimore groove"; that is, one lug has a groove or recess in it which provides a low resistance contact for the refinery tank-room connections. In order to make this lug, a cast-iron block (see Fig. 6) is inserted in the open lug of the mold. This block is removed just before the anode is raised off the mold by the pin-lifting device, and is put back after the anode has been lifted off the mold and placed in the bosh.

Mold wash is used to keep the anodes from sticking to the molds. The wash used is an emulsion of ground silica and water, and is applied just before the molds have passed the bosh. The ground silica has a fineness of 93 per cent through 200 mesh.

Between one and two per cent of the anodes made are scrapped, either because of overweight or underweight or because of a defective "Baltimore groove." If the molds or the copper are cold, the "Baltimore groove" lug is liable to be incorrectly formed.

When new molds are put on the machine, "warmers" are made for the first two rounds of the machine; that is, about ½ in. of copper is poured into the mold to thoroughly dry and warm it. These "warmers" are scrapped.

Table 1 shows two complete analyses of anodes made. The first sample—charge No. 6—was taken when the smelter was treating some custom material high in impurities. Charge No. 13 was taken when only ores from the Horne mine were being smelted.

TABLE 1.—*Analyses of Anodes*

	Charge No. 6, Per Cent	Charge No. 13, Per Cent
Copper	99.4200	99.5000
Oxygen	0.0870	0.0940
Lead	Trace	Trace
Arsenic	0.0520	0.0004
Antimony	0.0018	Trace
Bismuth	0.0004	0.0003
Selenium	0.1930	0.2380
Tellurium	0.0430	0.0520
Iron	0.0068	0.0160
Nickel	0.0368	0.0204
Sulfur	0.0050	0.0030
Silver	0.1121	0.0456
Gold	0.0291	0.0290
	99.9870	99.9987

The average fuel consumption in 1933 to date, casting about five or six times a week, is about 90 lb. of coal per ton of good anodes made. When the furnace is operated at capacity, the fuel ratio is about 75 lb. per ton of good anodes. With lower production more coal is needed in order to heat the furnace after a layover.

The casting crew consists of 15 men, 10 on the furnace and casting machine and 5 on the inspection, trimming and loading of the anodes. The making of molds requires two men part time.

STARTING UP

Construction work was completed in December, 1930, and toward the end of January, when the refinery was ready for anodes, the furnace was started. Two inches of silica sand was spread over the bottom and fused in. The bottom was seasoned by charging and melting scrap copper

until the bottom would absorb no more. The first charge of about 120 tons of molten blister copper was refined and cast on Jan. 20, 1931. At the start most of the difficulties were due to: (1) inexperience of the furnace crew; (2) mechanical troubles with the pin-lifting device, owing to the piston being too small; (3) failure to make sufficient allowance for shrinkage of the copper in the mold; (4) jamming of the lifting pins, because the molds were not anchored to the revolving table. These were overcome by installing an air cylinder of 10-in. diameter on the pin-lifting device in place of the 6-in. diameter cylinder; by allowing more for shrinkage and by fastening the molds securely to the table.

About two weeks after the furnace had been started, it was noticed that the surface of the silica-brick bottom was becoming uneven, the joints being deeply worn, especially under the launder where the hot blister copper enters the furnace. Shortly afterwards, a decided hump formed in the bottom, and finally, on the morning of Feb. 28, while the charge was being poled, part of the brick bottom came up. The bricks were skimmed off, the charge finished and then cast. It was found that about half the bottom, between the bridge wall and the tap-hole, had come up, and since the tapping slot extended down only as far as the upper surface of the top course of bricks in the bottom, a pool of copper about 15 in. deep was left. As much as possible of this copper was dipped out with long-handled ladles, the temperature of the furnace being maintained throughout the work, which took about 48 hr. The furnace was then shut down.

The copper had penetrated to the top of the bottom course of fireclay brick, and between each course of brick was a sheet of copper from ⅛ to 1 in. thick. Removal of the brick and the copper was difficult work, but it was accomplished with the help of a crew of miners using jack-hammer drills with special wedge-shaped chisels (with the cutting edge wider than the shank). The sheets of copper were cut into pieces that could be handled, part of the furnace arch was removed, and the copper pulled out by the converter cranes. In the removal of the old bottom, the side walls were badly damaged, but they were merely shored up and left until the new bottom was put in.

But for the able work of Mr. Henderson, mine superintendent, and his miners, the whole furnace might have been destroyed before the copper was taken out. After all the copper had been dipped out of the bottom, there was left only one 4½-in. course of fireclay brick, the other 9-in. course of fireclay brick and the 9 and 15-in. courses of silica brick being destroyed.

Monolithic Bottom

About this time the writers were in Ottawa and had the opportunity of discussing the question of a new bottom with Mr. F. E. Lathe, of the

National Research Council of Canada, and Mr. D. W. Stewart, of Canadian Refractories, Ltd. Mr. Lathe and Mr. Stewart have taken part in the development of a basic refractory made from Quebec magnesite and Mr. Stewart has had experience in using this refractory in basic open-hearth steel furnaces. The question of the feasibility of installing a monolithic basic hearth in a copper furnace was discussed in detail, and it was agreed that there were no theoretical objections or practical difficulties apparent. With the consent of Mr. E. Hibbert, consulting engineer, it was decided to make the experiment.

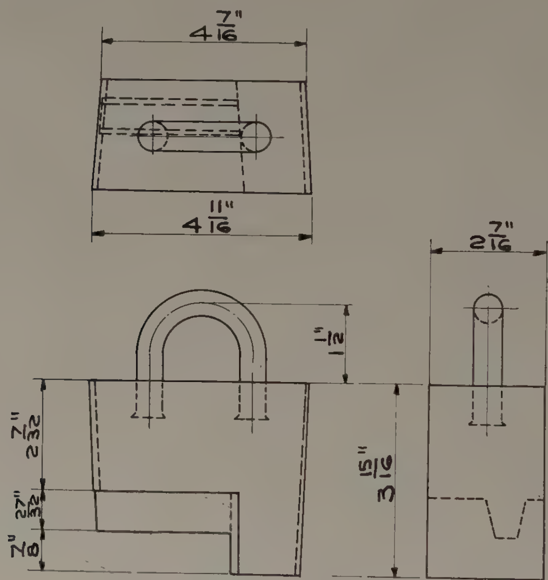


FIG. 6.—INSERT BLOCK FOR MOLDS.

The material used in the bottom is known by the trade name "Magnifrit." It is basic in nature and consists chiefly of magnesia and lime, with small amounts of iron oxide, alumina and silica. Magnifrit is manufactured from deposits of dolomite which occur at Kilmar, Quebec. Several grades of this rock are blended to give the desired analysis. The material is ground, thoroughly mixed and then sintered in rotary kilns with pulverized coal. The resulting sinter is crushed and screened. The final product is a brown, porous material, uniform in size, with pieces varying from 1/4 to 3/8-in. diameter.

The work of installing the new bottom was done under the supervision of Mr. Stewart and Mr. Schaup, of Canadian Refractories, Limited.

As mentioned before, all the original brick in the bottom had been destroyed, except the 4 1/2-in. course of fireclay brick that was directly on the concrete base. Before starting to put in the basic monolithic bottom, a course of 9 in. of fireclay brick was laid on top of the 4 1/2-in. course,

thus making 13½ in. of fireclay brick on the concrete base. No other bricks were put in the bottom. Fireclay brick was used because it has very little expansion and therefore would make a better bottom for the fused Magnifrit. In order to insure a proper bond between the monolithic hearth and the magnesite side walls, some of the silica bricks were removed form underneath the magnesite bricks of the side walls.

The first operation before beginning the fusion of the Magnifrit was to spread a parting layer of iron scale over the fireclay brick. The furnace temperature was then raised to 2400° F. and the layer of iron scale was fused in place. The furnace temperature was increased again and about 2500 lb. of Magnifrit, with a small amount of iron compound, was mixed and spread in a thin even layer over the bottom of the furnace. Care was taken that the two materials were thoroughly mixed before charging. The mixture was thrown in with shovels through openings in the sides and front, as rapidly as possible in order to prevent undue cooling of the furnace. The openings were then closed and the furnace temperature maintained at about 2700° F. for several hours until the addition was fused. Successive layers were added in the same manner. The log of putting in the bottom is shown in Table 2. About 24 additions were made, in layers of from ½ to ¾ in. thick. About 3000 lb. of material was used in each application and additions were made at an average interval of 4 hours. The function of the iron compound, as will be explained later, was to reduce the fusion temperature of the Magnifrit and to act as an aid in bonding.

Before an addition was made, each layer was tested by taking a core sample by means of a tee and a nipple screwed on to the end of a length of pipe. If the layer was correctly fused, the sample had a cheeselike consistency, would be well bonded together, and have considerable strength when cold. No fresh addition was made until several such samples taken at various points on the bottom showed that the layer was correctly fused.

As the side walls of magnesite brick below the metal line had been badly shattered by the vibration of the air drills when cutting out the copper, pieces of magnesite brick fell off on to the bottom from time to time as it was being fused. When this happened, the fires were checked slightly and as much of the brick as possible removed with flat steel paddles.

The top layer was surfaced with a mixture of iron compound and Magnifrit cement to give a smooth, hard surface. The finished bottom was 19½ in. thick in the center; in cross-section it was concave and in longitudinal section it was sloped so that the lowest point was at the tapping slot.

In all, 28 tons of Magnifrit and 9 tons of iron compound were used, and the actual work of making the bottom took 108 hr. When the last

Table 2.—*Record and Log of Construction of Bottom of Furnace*

Date	Additions to Bottom		Material Used, Lb.		Furnace Temperature, Deg. F.	Operation and Results
	Started	Finished	Magnifrit	Iron Compound		
March 13	7.00 a.m.		1,320		2,300	
March 14		9.00 a.m.				Small bath, ash, slag in center, used spoon at front end. Spread all over with slight bank to fill cracks.
	9.00 a.m.	10.45 a.m.	1,450	1,960	2,300	Soft in center.
	11.15 a.m.	11.30 a.m.	600	300		
	2.25 p.m.	2.50 p.m.	1,370	1,030	2,650	Well sintered over whole bottom. Two samples show good bond.
	3.55 p.m.	4.15 p.m.	1,220	610	2,750	Again well sintered; banks and corners well fritted.
	7.55 p.m.	8.15 p.m.	1,800	800		Not so well sintered but bottom hard. Not soft but heat a little low—more coal.
March 15	12.30 a.m.	12.45 a.m.	1,800	620		Furnace good heat and bottom well fused. Air cut, coal high but roof good. Overhanging brick side wall gave way, falling out.
	3.15 a.m.					Side wall on two sides gave way. Mr. Boggs and Mr. Schaup decided to bank up walls where overhanging and burn in for long period. Roof O.K. Banks at front very hot.
	4.30 a.m.	6.30 a.m.	3,000	750	2,800	Walls banked, probably 14 in. high; 4:1 mix banking walls, repairing. Fires on at 10.30 a.m.
	7.00 a.m.	10.30 a.m.	5,100	1,750		
	4.30 p.m.	4.45 p.m.	1,800	600		No. 5841—from back end—good core (12.20 p.m.).
	8.20 p.m.	8.35 p.m.	2,400	800		
March 16	12.20 a.m.	12.35 a.m.	2,250			
March 16	4.10 a.m.	4.25 a.m.	2,400	670		Good heat.
	9.40 a.m.	10.05 a.m.	2,250	450		Good heat.
	1.25 p.m.	1.45 p.m.	2,250	450		Bottom well sintered—good heat.
	5.50 p.m.	6.05 p.m.	1,800	300		Fireclay brick falling from flue. Good fusion.
	9.20 p.m.	9.35 p.m.	2,400	800	2,750	
March 17	12.20 a.m.	12.35 a.m.	2,700	600		
	4.50 a.m.	5.05 a.m.	3,150	600		
	8.45 a.m.	10.30 a.m.	2,100			Cooled furnace to remove brick from side wall. Door arch with rabbles, No. 5854. Good front end. Good sample.
	2.25 p.m.	2.50 p.m.	1,800			
	6.40 p.m.	7.00 p.m.	2,700			
	10.45 p.m.	11.00 p.m.	2,700	600		
March 18	5.00 a.m.	5.15 a.m.	2,400	600		Not so well fritted.
	10.30 a.m.	11.15 a.m.	2,700	650	2,750	
					Magnetite Cement	
March 18	2.15 p.m.	2.45 p.m.	1,800	450	450	
	8.00 p.m.	8.30 p.m.		1,200	2,400	
	11.15 p.m.	11.30 p.m.		1,200		For slagging.
March 19	5.00 a.m.					Fire off at 5.00 a.m.
	7.50 a.m.					Patching side walls and bridge wall, about 3½ tons material used.
March 19	11.30 a.m.					Fire on again.
	2.30 p.m.	2.45 p.m.				Charging about 2 tons scrap copper.
	3.35 p.m.					Copper all melted. Small pools evident. Much slag, probably copper oxide and magnesia.
	4.20 p.m.					Charged about 3 tons scrap copper.
	5.30 p.m.					Charged about 2 tons scrap copper.
	6.30 p.m.					Charged about 3⅓ ladles hot blister.
	9.00 p.m.					Tapped furnace without refining copper.
	11.30 p.m.					Furnace empty. Bottom good shape.
March 20	1.00 a.m.					Started patching side walls. Furnace better shape.
	10.20 a.m.					Started charging furnace 7 ladles copper.
			55,940	17,790	2,850	

addition had been fused, the fires were cut off and the furnace was slowly cooled to a red heat. The bottom was then seasoned in the same way as the original silica-brick bottom. After this had been done, about 70 tons of liquid blister copper was charged into the furnace. The copper bubbled, and it was assumed that this was due to further absorption of copper by the bottom. No attempt was made to refine this charge and it was merely cast into anode shapes, which were subsequently remelted. The following day a full charge of 130 tons of liquid copper was put into the furnace and normal operations were resumed.

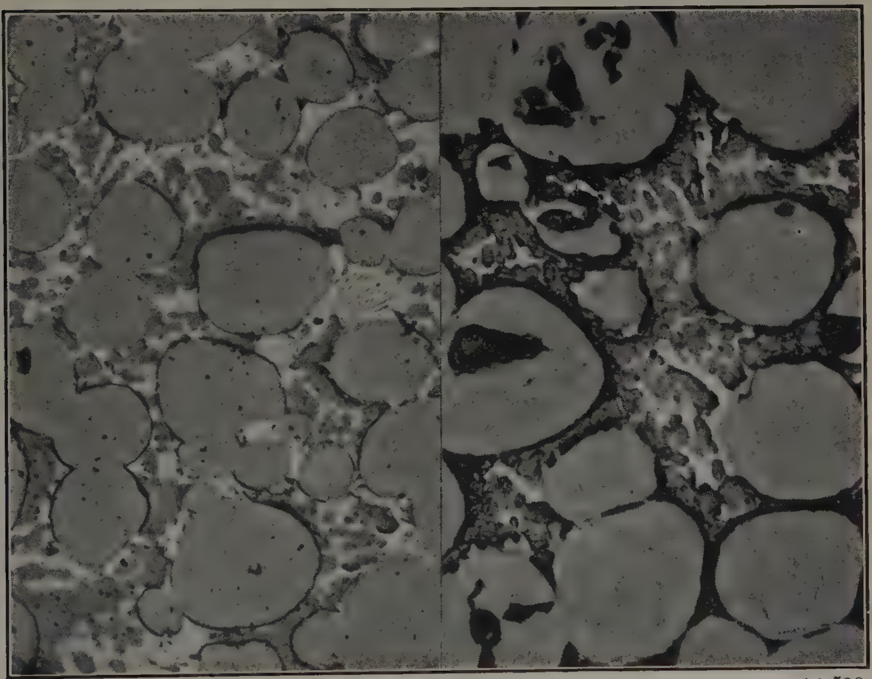


Fig. 7.—Photomicrographs of polished sections of core samples. × 500. Large rounded grains, periclase (MgO); smaller irregular darker grains, Ca-Mg-Fe silicates; white matrix, solid solution of Mg-Ca ferrites.

Success in installing such a bottom lies in spreading the material in thin, even layers, and in fusing each layer to a semiplastic condition before adding more material. If this is done, each layer is fused to the layer immediately beneath and the resulting bottom is a true monolith. If any bonding material is used, it should be thoroughly mixed with the refractory and the smallest quantity necessary should be added. Had it been possible to maintain in the furnace a sufficiently high temperature to bring straight Magnifrit to the point of incipient fusion, it would not have been necessary to use any iron compound. What was actually done in operation was to endeavor to use the minimum amount of iron

compound necessary to suit the temperature conditions in the furnace at the time of the addition of each layer.

Photomicrographs have been made of polished sections of some of the core samples. Two typical photographs, taken with vertical illumination at a magnification of 500 diameters, are shown in Fig. 7. The mineralogical character of the various constituents of the samples, shown by these photographs, has not yet been definitely settled, but the description that follows is in accordance with the best information available at the present time. There are three constituents. The large rounded grains are periclase; that is, magnesium oxide, probably with small amounts of iron oxide in solid solution. The smaller, irregularly shaped portions, darker in color, are believed to be calcium, magnesium and iron silicates of low silicate degree, and it is probable that these silicates contain ferrites and possibly periclase in solid solution. The white matrix is a solid solution of magnesium and calcium ferrites. The uniformity of the size and shape of the periclase crystals and the distribution of the other constituents give a well bonded material, which will afford a maximum resistance to disintegrating forces.

About five weeks after the new bottom had been put in, it became necessary to rebuild the side walls above the slag line, as they had been badly damaged during the fluxing of the bottom. Unburned magnesite brick were used, faced on the outside with silica brick. When the furnace was shut down for this repair, a hump was found in the bottom at the front end of the furnace. An attempt was made to drill this off, but it resisted such efforts. In June, a section of the bottom came up at this point, leaving a hole several feet in diameter, and from 3 to 4 in. deep. The copper was dipped out of the hole, which was patched by fusing in thin layers of material. This patch was not successful and gradually came up, so that in August it was necessary to patch the bottom again. The failure of the first patch was due to the fact that the work had been done at night, and in the imperfect light the operators had mistaken grain magnesite for Magnifrit. In August, 1931, a second patch was put in with Magnifrit. During the spring of 1933 some of the patch came up. In June, 1933, some high-grade silver concentrates containing many impurities, particularly sulfur, were treated in the anode furnace, and it is believed that excessive work done on these impure concentrates caused the hole in the bottom to become enlarged, and this part of the bottom was then below the tap-hole. On Nov. 2, 1933, this hole was dipped out clean and new layers of a mixture of Magnifrit and iron compound were fused in place, so that the whole bottom was raised above the level of the tapping slot. The bottom was seasoned with copper, as before, and the furnace resumed normal operation.

Since the monolithic bottom was installed, the following work has been done and the results indicated have been obtained.

1. March 19, 1931 to June 1, 1931, a campaign of 74 days. A small patch requiring 1000 lb. of material was fused in.
2. June 2, 1931 to Aug. 26, 1931, a campaign that lasted 85 days. Only a small patch was made and the job was completed in 16 hours.
3. Aug. 27, 1931 to April 9, 1932, a campaign of 227 days. This was a larger patch and required three days to complete.
4. April 12, 1932 to Nov. 1, 1933, a campaign of 568 days. A large patch was made at this time, requiring about 5 tons of Magnifrit and ½ ton of iron compound. Four days were required to make the patch.
5. In all, 86,415 tons of copper were taken out of the furnace from March 19, 1931, to Nov. 1, 1933.

From past experience, the bottom, at time of writing (November, 1933) should give continuous service for several years without requiring repairs or patching.

So far as the authors know, this is the first successful application of a basic monolithic bottom for use in a copper-refining furnace. It is an improvement over either silica or magnesite-brick bottoms, which must be built and expanded by experts. With a brick bottom, great care must be taken that the bricks are not damaged in any way, because, if they fail at one point, the whole bottom may be destroyed, with a resulting extended shutdown and expensive repair work. An inherent advantage of a monolithic bottom is that it can be patched readily and cheaply at the convenience of the operators and with very little loss of time. It is believed that the failure of the original silica brick was due to the cutting action of the molten stream of copper falling from the launder through which it was poured into the furnace.

Because this type of bottom can be patched successfully, and because it is in good condition after about three years of operation, the indications are that it will have a useful life of many years.

Montreal East Plant of Canadian Copper Refiners Limited

BY H. S. MCKNIGHT,* MONTREAL EAST, QUEBEC, CANADA

(New York Meeting, February, 1934)

ONE phase of Canada's rapid development in mining and metallurgy during the past few years has been the creation of Canadian Copper Refiners Ltd., and the establishment of its modern electrolytic copper refinery at Montreal East, Quebec. Canadian Copper Refiners Ltd. is made up of three well-known metal firms whose combined activities cover the entire industry from the mine to the market. They are Noranda Mines Ltd., Toronto, Nichols Copper Co., New York, and British Metal Corporation, Ltd., London. Noranda Mines is the principal owner and provides the greater part of the raw material received by the refinery for treatment. Nichols Copper Co. designed and built the plant and is operating it under contract. British Metal Corporation acts as sales agent and markets the refinery's products.

Montreal East was chosen as the site of the new refinery only after a careful survey had been made of the entire eastern part of Canada. The location chosen offers many advantages, chief of which are the relatively low costs of power, labor, fuel and supplies, and the fact that it is situated on Montreal harbor, the nearest seaport to the points of origin of the raw materials.

All incoming material is received by rail. A majority of the plant's product is exported through the port of Montreal to European markets, being handled to the docks in railroad cars. A large proportion of the product is delivered by means of an industrial railway to the adjacent rolling mill of the Canada Wire & Cable Co., Limited.

The refinery was designed to treat primarily the copper produced by Noranda Mines, Ltd. and Hudson Bay Mining & Smelting Co., Ltd., and its rated capacity is 75,000 short tons per year. A general layout of the plant is shown in Fig. 1. Its capacity can be readily doubled, as shown by the dotted lines. Figs. 2 and 3 are sections taken through the furnace department and the tank house respectively. Except for the main office and laboratory and an auxiliary building containing the stores, warehouse, change rooms, compressor room and machine shop, the entire plant is under one roof. The type of building construction consists of reinforced concrete foundations supported on concrete piles, a steel structure covered with Robertson Protected Metal siding and a 3-in.

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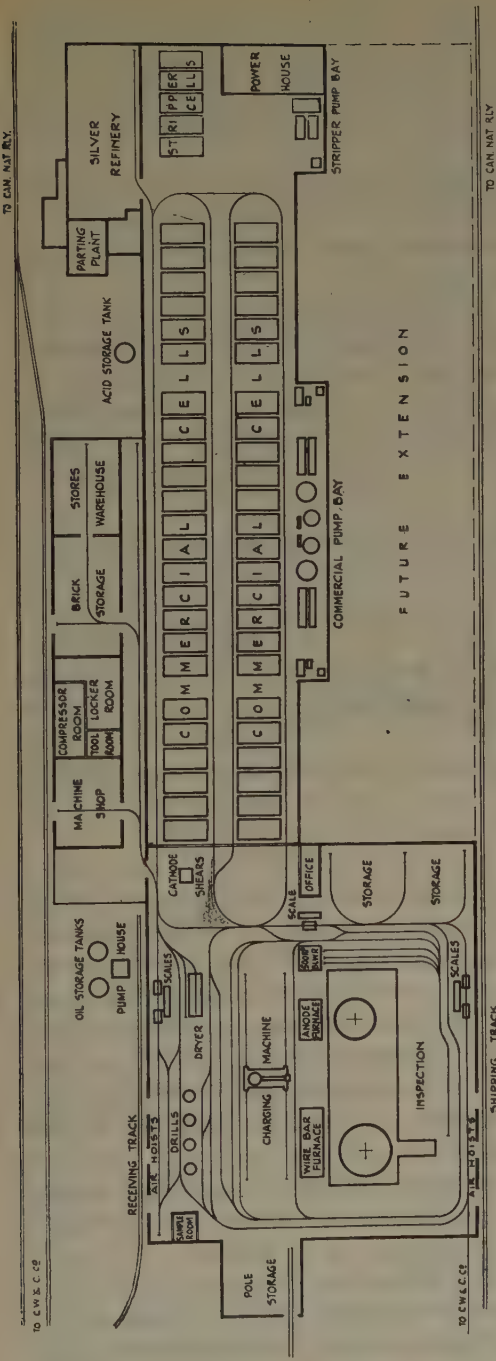

FIG. 1.—GENERAL LAYOUT OF PLANT.

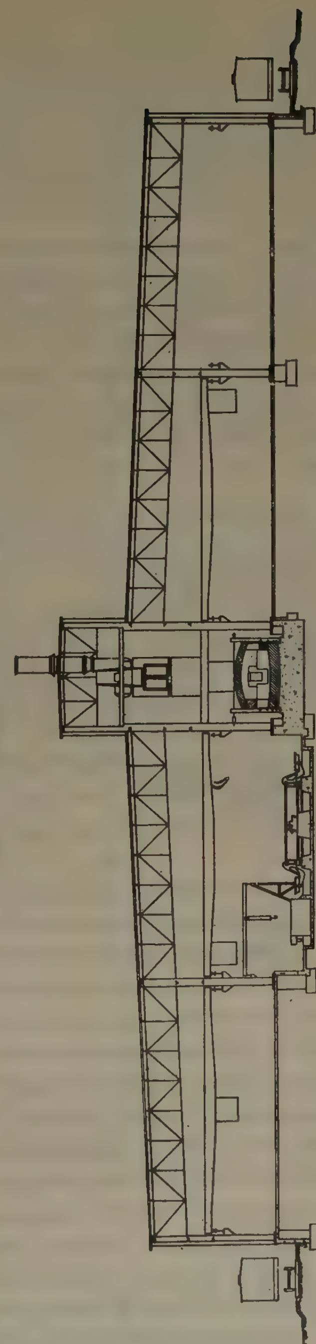

FIG. 2.—SECTION THROUGH FURNACE BUILDING.

thick wood roof deck. The roof deck is covered with a standard 15-year built-up tar and gravel roof. The roof is designed with alternating high and low bays so that abundant light is provided throughout the building from above by means of monitor type windows. The power house and the parting plant are built of brick.

The building is heated by unit high-pressure steam heaters along the walls. Air is drawn from outside or inside the building and forced through the steam coils by motor-driven blowers. Foul air is discharged through stationary ventilators in the roof.

The working floors throughout the plant are at one elevation, this being approximately 4 ft. above ground level. Materials are handled between departments by means of 30-in. gage industrial cars drawn by two 10-ton Westinghouse storage-battery locomotives. Auxiliary locomotive power is furnished by a Fordson tractor equipped with smooth hard rubber tires. The cars are of two types: (1) anode rack cars built

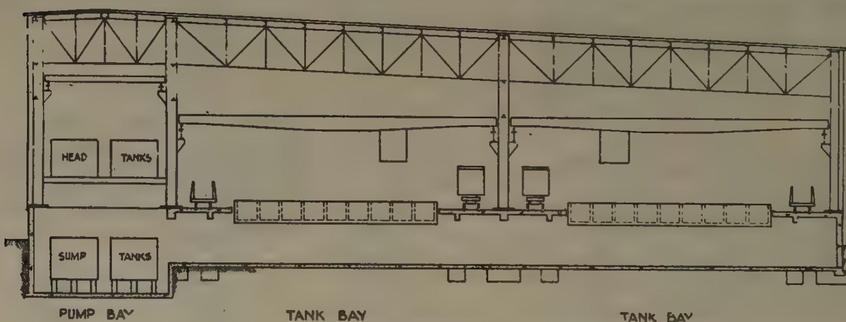

FIG. 3.—SECTION THROUGH TANK HOUSE.

to handle 21 anodes and equipped with double trucks, cast-iron wheels with chilled steel tread and Timken roller bearings; (2) cathode cars of the single truck design with chilled steel treads and Hyatt roller bearings.

A private pumping station delivers from the St. Lawrence River all water required for process work. The station is equipped with motor-driven centrifugal pumps having a combined capacity of 4500 gal. per minute. Water for drinking and bathing purposes is obtained from the Montreal city water system. Water for process work may also be had from this source in case of necessity.

Bunker C fuel oil is used for firing the copper furnaces and boilers. The oil is delivered to storage tanks having a capacity of 45,000 gal., through pipe lines directly from a neighboring oil refinery. The oil is delivered from storage to the burners through a circulating system at a temperature of 200° F. and a pressure of from 35 to 50 lb. per square inch.

Incoming material is received in the form of anodes from Noranda Mines and blister cake from Hudson Bay Mining & Smelting Co. More than 60 per cent of the total material received is from Noranda. These

anodes are 36 in. square on the face, 1½ in. thick and weigh approximately 700 lb. The Hudson Bay blister, which is received in cakes weighing approximately 450 lb., is combined with the anode scrap from the tank house in the anode furnace. From this material domestic anodes are made, which have the same physical characteristics as the Noranda anodes but somewhat different chemical composition. Typical analyses of the two anodes entering the tank house are given in Table 1.

TABLE 1.—*Typical Analyses of Anodes*

Anode	Cu, Per Cent	Ag, Oz. per Ton	Au, Oz. per Ton	As, Per Cent	Sb, Per Cent	SeTe, Per Cent	Pb, Per Cent	Ni, Per Cent	Fe, Per Cent	S, Per Cent
Noranda	99.41	17.09	8.058	0.0024	0.0023	0.264	0.0016	0.0293	0.0065	0.0140
Domestic	99.49	31.40	4.577	0.0126	0.0073	0.183	0.0236	0.0143	0.0083	0.0155

RECEIVING AND SAMPLING

Because of its high content of precious metals, all incoming material is weighed and sampled with extreme care. The material is received in box cars at the receiving bay (Fig. 1) on the east side of the building, each carload constituting a lot. The anodes are received in lots of 168 pieces and the blister in lots of 221 pieces. After the material has been unloaded from the box cars, it is placed on industrial cars and weighed and check-weighed. One small carload from each lot, taken from near the center of the string, is placed in a drying oven for 24 hr. to determine the moisture content. Each small car holds from 7 to 9 per cent of the total lot.

After weighing, each piece of every lot is sample-drilled according to a template. For anodes, a quarter template is used and all holes are drilled from the set side. It requires four lots to have the entire face of the anode covered by the template. For blister cakes, a half template is used, requiring two lots to have the entire face covered by the template. In the blister cakes all holes are drilled from the bottom, or mold side, except the edge holes, which are drilled from the top. The drilling is done on motor-driven drill presses having automatic feed and using ½-in. diameter drills. All drillings from each lot are combined, dried, ground and divided into necessary samples for analysis. After weighing and sampling, the anodes are placed in rack cars and delivered to the electrolytic department. The blister cakes are placed on cathode cars and delivered to the anode furnace department.

TANK ROOM

The tank room was designed to operate on a normal current density of 17 amp. per square foot of depositing area and with a 4½-in. spacing from center to center of anodes. The starting sheets are 37½ in. square and, as stated above, the effective area of the anode face is 36 by 36 inches.

There are 432 commercial cells and 36 stripper cells. The commercial cells are 16 ft. 7 in. by 3 ft. 7½ in. by 4 ft. 1½ in. deep inside and are built of reinforced concrete cast in place and lined with 8 lb. 6 per cent antimonial lead. The cells are built in tiers of nine cells to a tier as shown in Fig. 4 and arranged in sections of two tiers, or 18 cells, to a section.

The cells are supported independent of the working floor by concrete columns and at an elevation such that the tops of the cells are 18 in.

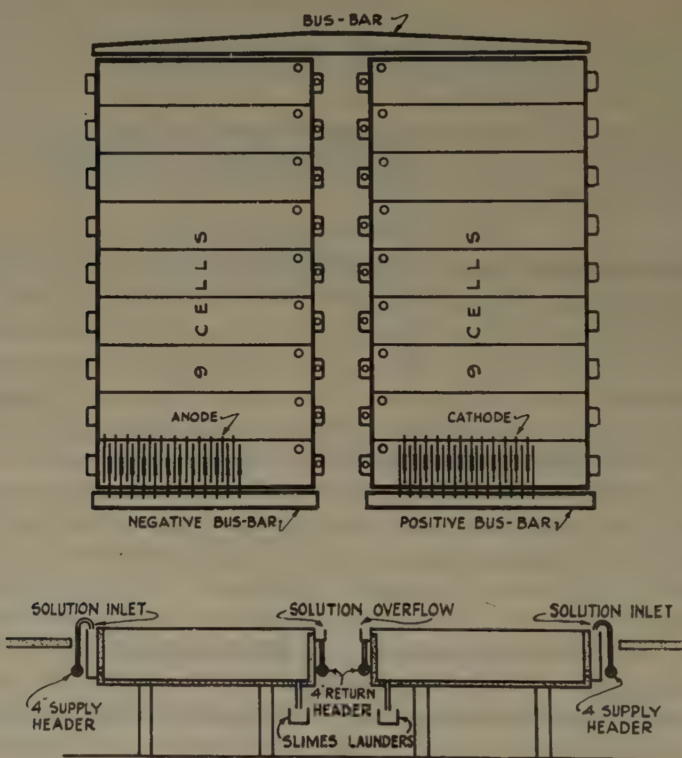

FIG. 4.—CELL TIERS.

above the floor. The columns are capped each with a glass plate 1½ in. thick, a rubber sheet and a lead shield to insulate the cells from the ground. The bottoms of the cells are 7 ft. above the basement floor, thereby providing ample space for inspection and headroom for solution lines and slimes launders.

The tank room is 660 ft. long and 144 ft. wide (Fig. 5). The commercial and stripper cells are in two 60-ft. parallel bays and the electrolyte heaters and circulating pumps and tanks are in a third 24-ft. pump bay west of the commercial bays. The stripper circulating system, of course, is independent of the commercial system. The electrolyte

flows by gravity from head tanks through the electrolytic cells to sump tanks in the pump bay located below the basement floor. It is then elevated by means of motor-driven Corrosiron centrifugal pumps to the

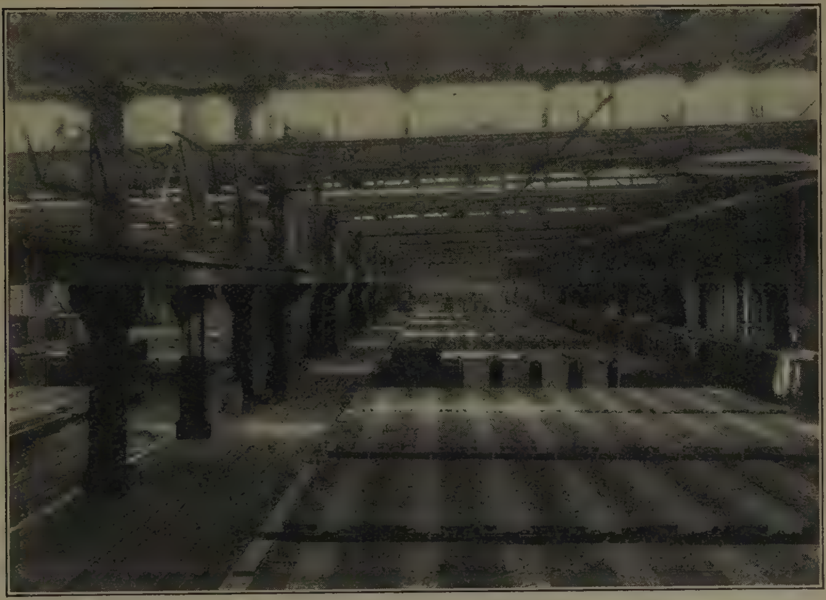

FIG. 5.—TANK ROOM.

head tanks, where it is heated by steam coils. The temperature of the electrolyte is maintained at approximately 140° F. and the flow through each cell is at the rate of from 4.5 to 5 gal. per minute.

Table 2 shows typical analyses of the two electrolytes.

TABLE 2.—*Typical Analyses of Electrolytes*

Electrolyte	Temperature, Deg. F.	Deg. Bé at 60° F.	Free H_2SO_4, Per Cent	Cu, Per Cent	Specific Gravity	Cl, Per Cent
Stripper	140	22.50	12.96	3.16	1.19	0.00075
Commercial	140	27.30	16.60	3.20	1.24	0.00019

The inlet to each cell is of rubber to prevent loss of current through the solution lines. The overflow line is broken by an air gap in a lead "boot."

The initial loading to each cell is 42 anodes at 4½-in. spacing and 43 starting sheets. As before stated, the anodes weigh approximately 700 lb. each, so that, operating at a current density of 17 amp. per square foot, the anode life is 33 days. Two cathodes are made from each

anode. When the first cathode is pulled the anode spacing is closed up to 4 in. and five additional anodes added to each cell. This lowers the current density during the second cathode run, with a consequent power saving.

The commercial and stripper cells are all connected in series in one electrical circuit but the nine cells of each tier form a series-parallel connection. The bus current of 15,000 amp. flows into each cell through the 42 anodes and out at the other side through the 43 cathodes. The anodes are cast with a "Baltimore groove" in the top of one of the lugs to form an electrical connection between the anode of one cell and the cathode of the adjoining cell.

A small test switchboard at the foreman's office is connected to each section of tanks in such way that the voltage drop across any section can be read at any time. Both in the construction and in the operation of the plant great care has been taken to eliminate and prevent power losses through short circuits, and results obtained have been excellent. The normal bus current is 15,000 amp. and the average voltage drop across each commercial cell while operating at a current density of 17 amp. is 212 mv. The current density in the bus is approximately 375 amp. per square inch.

About 300 tons of copper was used in the bus system which is 3500 ft. long. The main runs under the working floor are made up of eight 10 by ½-in. copper bars spaced ½ in. apart in parallel. The tank buses are of different sizes and shapes, the largest being 17 in. wide, 3 in. thick and 19 ft. long. The tank buses are connected to the main run by two 10 by 2-in. U-connector bars. The current is transmitted from the bus to anode and from cathode to bus through knife-edge contacts. These contacts consist of triangular sections of copper arc-welded to the 3-in. bus bars. About 4000 lineal feet of arc welding was required and a highly satisfactory joint was obtained.

The anodes are delivered to the tank room in rack cars. The rack cars contain 21 anodes each, or half a cell load. The spacing of the anodes in the cars is the same as required in the cells, and they are transferred from the cars to the cells by overhead electric cranes. These cranes are of the three-motor overhead trolley type and are driven by 550-volt a.c. motors with face-plate controllers located above the operator's head and operated by vertical levers.

The starting sheets are made in the stripper cells from anodes slightly larger than the commercial anodes and weighing approximately 770 lb. each. They are deposited on cold-rolled copper sheets from which they are stripped daily after they have attained a thickness of approximately 1/32 in. The average weight of a starting sheet is 11 lb. The starting sheet is attached to the suspension rod by means of a single central loop 18 in. long riveted to the sheet.

The cathodes are removed from the electrolytic cells in racks handling a full cell load of 43 pieces. They are transferred by the overhead cranes to movable washing machines, one in each of the two 60-ft. bays. These washing machines are mounted on wheels, so that they can be operated next to the sections of cathodes being pulled, thereby avoiding the necessity of transporting the dripping cathodes over other sections. The wash water is supplied to the machines at a pressure of from 50 to 65 lb. per square inch through a main in the center of the building, by means of Corrosiron two-stage centrifugal pumps. The wash solutions flow back from the washing machines to pump tanks in the center of the building.

The slimes are withdrawn from the cells at the same time that the cathodes are removed; that is, twice during the anode life. At the end of the first cathode run the slimes are removed but the anodes are not moved. At the end of the second cathode run the anode scrap is also removed and the tanks are completely emptied and cleaned. The slimes removal is accomplished through a plug opening in the bottom and at one end of each of the electrolytic cells. First, the clear electrolyte is pumped from the cell to within a few inches of the bottom. The plug is then removed and the slimes, together with the remaining electrolyte, are swept through the plug opening into a lead-lined slimes launder beneath the tiers of cells. These launders convey the slimes-bearing solution to centralized pump tanks on the center line of the building and in open concrete pits below the basement floor. From these tanks they are pumped to a holding tank in the silver refinery by means of specially designed Corrosiron centrifugal pumps. The slimes carry the precious metals and impurities contained in the anodes. The further treatment of these slimes will be described later in the paper.

After passing through the washing machines and having the electrolyte thoroughly removed, the cathodes are delivered on cathode cars to the wire-bar furnace department. The anode scrap is washed in the electrolytic cells by means of high-pressure water sprays. After washing, it is loaded on cathode cars and delivered to the anode furnace department. Approximately 14 per cent of the anodes is returned as scrap.

Furnace Department

The copper furnace equipment consists of one anode and one wire-bar furnace in the central 24-ft. bay of the furnace building as shown in Fig. 1. The two furnaces are of the usual reverberatory type with side charging doors and fired each by means of a single high-pressure oil burner. Atomization at the burners is done by either air or steam, as desired, and secondary air is supplied by motor-driven blowers mounted on the floor under the burner stands. Furnace draft is supplied by motor-driven induced-draft fans. Draft regulation is obtained by

regulating the speed of the fans and also by means of dampers in the flues.

The furnaces are built on solid concrete foundations, the tops of which are shaped to conform with the inverted arch of the furnace bottoms. The bottoms are made up of three courses of brick laid directly on the concrete foundations. The lower course is of 12-in. clay brick, the second course of 9-in. silica brick and the third, or top course, of 15-in. silica brick. The side walls below the metal line are built of burned magnesite brick. The bridge wall, side walls above the metal line, and the "nose" of each furnace are built of magnesite brick backed up with clay brick. The verb arch and main roof are built of silica brick. The firebox and uptake are built of clay brick.

Both furnaces are charged by means of a six-motor revolving floor-type Morgan charger having a floor span of 36 ft. and a peel stroke of 24 ft. 6 in. The load capacity of this machine is 8000 pounds.

Anode Furnace

The anode furnace has a capacity of 150 tons per charge. The hearth is 26 ft. 9 in. by 12 ft. 8 in. and the uptake is 2 ft. 1 in. by 5 ft. 0 in. There are two charging doors on the east side of the furnace. These doors are 6 ft. 0 in. by 3 ft. 3 in. and are operated by air cylinders on the floor above. The blowing is done by introducing through holes in the charging doors ¾-in. iron pipes to which are attached air hoses with 30-lb. air pressure. The skimming and poling are done through the front door. Slag and poles are handled by means of a 5-ton floor-controlled electric hoist on a trolley.

Directly over the uptake from the furnace is a 250-hp. straight-tube waste-heat boiler equipped with a superheater to deliver 100° superheat at 150 lb. working pressure. The induced-draft fan and dampers are beyond the boiler. The fan is built to deliver 36,000 cu. ft. of gas per minute against a static pressure of 3½ in. and at a temperature of 650° F.

The casting wheel (Fig. 6) is in a pit west of the furnace. This pit consists of a depressed 60-ft. bay of the furnace building and extends 180 ft. alongside of the two furnaces. It contains all the casting equipment for both furnaces and is served by a 10-ton overhead crane similar to those serving other parts of the plant. The wheel is 33 ft. in diameter and is of the Walker type. It is motor-driven through a Waterbury hydraulic variable speed transmission with gear control. The anode molds are made of copper and are mounted on a stationary table on the wheel. The movement of the wheel is intermittent, the copper being poured from the ladle into the molds while the molds are stationary. The molds are sprayed with water as they move towards the bosh, which is diametrically opposite the tap-hole. The anodes are raised by lift pins extending through the bottoms of the molds, so that they can

be transferred from the molds to the bosh by means of a hand-operated airlift.

The bosh is made up of two compartments, each having spacing racks to hold an anode rack-car load of 21 anodes. When one compartment has been filled its load is transferred to a rack-car by the overhead crane

Fig. 6.—Casting wheel.

while the second compartment is being filled. The average rate of casting is about 45 tons per hour.

In casting anodes, pulverized silica is used as a mold wash. The anode molds are cast in collapsible cast-iron mother molds, using wire-bar copper. The molten copper is transferred from the wire-bar furnace tap-hole launder to the mother molds in a geared crane ladle.

The anode ladle is operated by a suspension-type electrically driven hoist. Both the ladle and the wheel are controlled by the ladler whose cab is located above the floor near the ladle and outside of the wheel.

Wire-bar Furnace

The wire-bar furnace has a capacity of 350 tons per charge. The hearth is 43 ft. 2 in. by 14 ft. 9 in. and the uptake is 2 ft. 8 in. by 7 ft. 8 in. There are three charging doors operated in the same way as those on the anode furnace. Blowing of the charge is done through holes in these doors and through blowing holes provided in the furnace side walls on the casting side. The method of handling slag and poles at the front door is the same as at the anode furnace.

The furnace is equipped with one 800-hp. cross-drum straight-tube waste-heat boiler with superheater and with two motor-driven variable-speed induced-draft fans.

The wire-bar casting wheel (Fig. 7) is 40 ft. in diameter and is of the Walker type. It is west of the furnace in the casting pit referred to above, and is driven by a constant-speed motor through a Waterbury variable-speed transmission. The movement of the wheel is controlled by the ladler whose cab is located above the wheel and directly in front

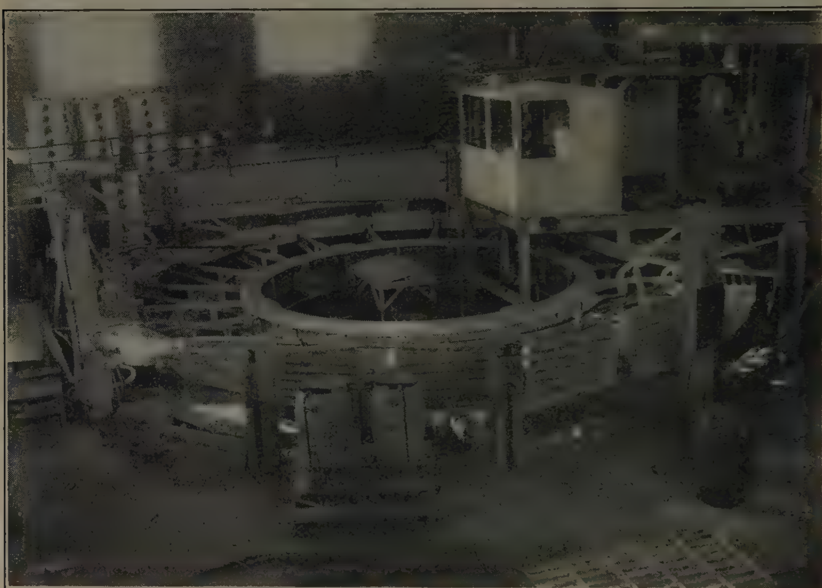

FIG. 7.—WIRE-BAR CASTING WHEEL.

of the spout of the ladle. The ladle control is similar to that on the anode wheel.

The molds are suspended on trunnions between arms placed radially on the ring of the wheel. The movement of the wheel during casting is continuous, the copper being poured into the mold pockets as they pass slowly under the spout of the ladle. After being filled, the molds pass over water sprays and are dumped by means of dumping rails over a bosh directly opposite the tap-hole.

After dumping, the molds are sprayed on the bottom to reduce the temperature and on the inside to remove any excess mold wash or dirt. The pockets are then sprayed with fresh mold wash consisting of a mixture of bone ash and water and are righted preparatory to being refilled. The molds are inspected just before they pass under the ladle, and the temperature is taken and recorded.

The wheel is equipped to cast all standard sizes of wire bars, ingot bars and ingots. Molds are made daily of wire-bar copper on a mold press

that is near the tap-hole. The stream of copper from the tap-hole is divided and part of it is carried through a launder to a collapsible cast-iron frame on a mold car under the press. Water-cooled copper dies mounted on the press are then lowered into the molten metal and allowed to stand until the metal is set. They are then raised and the mold removed to cool.

Each mold is numbered for recording purposes and is used for no more than 10 productions. The old molds are returned to the furnace and remelted. All wire-bar shapes are cast in 5-bar molds, and for the

FIG. 8.—COTTRELL PLANT.

larger bars 20 molds make up a full ring on the wheel. The casting rate is from 55 to 75 tons per hour, depending on the size of bars being cast.

Crucible samples are taken at the tap-hole at intervals of 30 min. during the casting operation. Optical oxygen determinations are made immediately and reported to the furnace foreman. Billet samples are also taken from the tap-hole during the run and conductivity tests made on wires drawn from these billets. Drill samples for complete chemical analysis are taken from the finished bars. The inspection department sets aside three bars during the cast—one at the beginning, one at the middle and one at the end of the cast. These bars are drilled and the drillings sent to the laboratory for analysis. A typical wire-bar analysis is as follows: Ag, 0.25 oz. per ton; Au, 0.015 oz. per ton; Cu, 99.965 per cent; As, 0.0001 per cent; Sb, 0.0001 per cent; Se and Te, 0.0002 per cent; Fe, 0.0016 per cent; Ni, 0.0002 per cent; S, 0.0013 per cent; Pb, 0.0007 per cent; O_2, 0.035 per cent; conductivity, 101.41.

The refinery brand "CCR" is stamped on one side of each wire bar before it is shipped. The bars are also stamped with the furnace production number and the inspector's number.

The wire bars are removed from the cooling bosh by means of a chain conveyor and are placed on inspection rails on either side of the bosh for inspection. Each bar is gaged and carefully examined for defects before being delivered to the transportation department for stocking or shipment.

All copper shipments are made from the west side of the building (Fig. 1). Double narrow-gage track scales are used for weighing all outgoing material and each shipment is check-weighed before loading into the railroad cars.

Silver Refinery

Under "Tank Room" was described the delivery of slimes-bearing solutions from the copper electrolytic tanks in the tank room to a holding tank in the silver refinery. From this holding tank the solutions are

Fig. 9.—Parting plant.

pumped at a uniform rate to a Dorr thickener. The overflow solutions from the thickener are returned through a settling tank to the tank room. The thickened slimes are filtered by means of an Oliver continuous filter, the filtrate being returned also to the tank room. The filtered slimes, containing approximately 25 per cent moisture, are discharged from the filter directly into a six-hearth 8-ft. diameter Nichols-Herreshoff roasting furnace, where they are roasted for approximately 1½ hr. to oxidize the contained copper and other impurities preparatory to leaching with acid. The roasting temperature ranges from 500 to 650° F. After roasting, the copper in the slimes is 97 per cent soluble in warm 20 per cent sulfuric acid.

The roasted slimes are transferred from the roasting furnace to lead-lined leaching tanks, where they are treated by agitating with 20 per cent acid solutions made up chiefly of foul electrolyte from the tank room. The temperature of the leach solution is approximately 190° F.

After leaching for approximately 3 hr., the insoluble residue is allowed to settle and the leach solutions are decanted off. The residue is washed

with water and then passed over a second Oliver continuous filter. The filtrate is returned to the wash solution head tanks and the filtered slimes delivered to the doré furnace.

The decanted leach solutions contain a considerable amount of copper and selenium in solution and require further treatment. From the leach tanks they are passed through a gravity filter to the purification head tank. The purification system consists of eight electrolytic cells employing lead anodes and lead cathodes. Practically all of the selenium and most of the copper are removed from the solutions in these cells, being precipitated in the form of a copper-selenium sludge. These sludges are stored for possible future treatment to recover the selenium. The solutions from the purification cells are treated in iron cementation tanks to remove the last traces of copper before being discarded.

Table 3 gives typical analyses of the raw, roasted and leached slimes.

TABLE 3.—*Typical Analyses of Slimes*

Slimes	Analysis, Per Cent											
	Cu	Ag	Au	As	Sb	Se	Te	Pb	Ni	Fe	SO_4	SiO_2
Raw	45.0	8.0	2.5	0.120	0.222	24.61	3.77	1.91	0.037	0.338	6.21	2.18
Roasted	42.0	7.5	2.3	0.083	0.107	19.17	3.18	3.77	0.014	0.358	5.61	2.17
Leached	3.8	31.0	9.3	0.076	0.242	24.62	2.43	10.16	0.006	1.030	5.38	6.56

The leached slimes are charged into a reverberatory doré furnace together with necessary fluxes of soda and niter for smelting and refining. A doré charge consists of approximately 8000 lb. of slimes containing 25 per cent moisture. From 36 to 48 hr. is required to complete the smelting and refining operations.

The refined doré bullion averages approximately 25 per cent Au, 74.5 per cent Ag and 0.5 per cent Cu and a representative furnace production weighs 40,000 troy ounces. The bullion is ladled by hand from the furnace and cast into anodes, which are 6 by 9 by ½ in. thick and weigh approximately 150 oz. each.

The first, or "scoria," slags produced from the doré furnace are treated in the copper anode furnace to recover the gold, silver and copper values. The soda and niter slags are leached with water and the residue returned to the doré furnace.

The doré anodes are delivered to the parting plant for the recovery of silver and gold. Here the parting is effected by means of Moebius electrolytic cells, which are of concrete lined with mastic, grouped in two units of five cells each. Suspended in each cell is a wooden frame or basket with a canvas-covered bottom in which the electrolytic silver crystals are collected. The silver crystals are scraped off the cathodes

by oscillating wooden scrapers operating from above. The cathodes consist of rolled silver sheets fastened by means of brackets to the busbars which in turn are carried by a removable wooden frame resting on the top of the cells. The anodes are suspended on removable hangers, in units of three, inside of muslin bags, which are placed in frames between the cathodes and are also supported by the busbars. There are 15 anodes per cell, or 150 anodes for a total plant charge. When all of the 10 cells are in operation they contain approximately 22,000 oz. of bullion. When the silver has been removed from the anodes by electrolysis the gold remains in the muslin bags in the form of a fine black mud. This gold mud is removed from the bags, washed with water and then boiled in a cast-iron pot with concentrated sulfuric acid until all traces of silver are removed. It is then washed and filtered and charged into a graphite crucible, where it is melted and cast into 500-oz. bars.

The silver is removed from the cells every 24 hr. by raising the baskets out of the electrolyte and scraping the crystals into tile wash carts. Here it is thoroughly washed with water and then charged into a graphite retort, where it is melted down and cast into 1000-oz. bars.

The gold bars are 992. + fine and the silver bars are 999. + fine. Cast-iron molds are used for casting both the gold and the silver bars.

The electrolyte in the Moebius cells is a nitric acid solution of approximately the following composition: Ag, 30 grams per liter; Cu, 20 grams per liter; free nitric acid, 7 grams per liter.

The flue gases from the doré furnace and from the Nichols-Herreshoff roasting furnace carry with them large quantities of precious metals. The amount of gas evolved from the doré furnace is approximately 13,000 cu. ft. per minute at 2500° F. That from the roasting furnace is approximately 800 cu. ft. per minute at 500° F.

These gases pass through brick and steel cooling flues, are mixed and then delivered to a three-tower scrubbing system at approximately 450° F. These scrubbing towers are equipped with a spray system through which water is continuously circulated. In this system approximately one-third of the solids is removed from the entering gases and the gases are fully saturated with water vapor.

From the scrubbers they pass to the Cottrell precipitator units at approximately 125° F. There are two Cottrell units in parallel consisting of twenty 8-in. diameter pipes each. The precipitator is of the pipe type and is made entirely of lead and lead-covered steel.

By means of damper controls the gas volume entering each unit can be carefully regulated. The collected precipitate is in the form of a wet sludge, which tends to slide off the collecting pipes and the discharge electrodes, but to insure maximum effectiveness at all times the pipes and electrodes are thoroughly washed down every day with a high-pressure hose.

The efficiency of the unit as indicated by tests is approximately 99.5 per cent and actual losses indicate silver at less than 1 oz. per day and gold at 0.03 oz. per day. During the first seven months of operation the sludge collected in the precipitator and in the scrubber system amounted to approximately 33,900 lb. and contained 34,900 oz. of silver and 8300 oz. of gold.

Power

All the electric power used in the plant is supplied by the Montreal Light, Heat & Power Consolidated at 12,000 volts, three-phase, 60 cycles from two different substations over two separate lines. One line is used as a reserve and can be instantly switched into service in case of trouble on the other circuit. The incoming 12,000-volt current is transformed to 2300 volts and 550 volts by two banks of outdoor transformers just south of the power house. The primary transformer bank consists of three 1500-kva. single-phase 12,000 to 2300-volt power transformers and the secondary bank consists of three 400-kva. 2300 to 575-volt transformers. Power is carried overhead from the outdoor substation to the main 2300-volt and 550-volt bus in the power house.

The power house, at the south end of the tank house, contains the control switchboard, switching equipment, motor-generator sets, storage battery and auxiliary equipment. The building is of brick construction, 80 ft. long, 45 ft. wide and 33 ft. high, with basement. Ventilation is provided by adjustable louvers admitting cool air to the main floor and basement while the heated air is discharged through ventilators in the roof.

The power for the main electrolytic circuit is supplied by three motor-generator sets consisting of 675-kw., 135 to 12-volt, 5000-amp. direct-current generators driven by 980-hp., 2300-volt, synchronous motors. The three sets operate in parallel, supplying 15,000 amp. to the circuit. The power for the purification circuit is supplied by a 125-kw., 25 to 3-volt, 5000-amp. direct-current generator driven by a 225-hp., 2300-volt synchronous motor. These sets are mounted on foundations independent of the building, to avoid transmission of vibration to the building structure.

The switching structure and equipment are located along the south wall of the power house and the main motor-generator units along the north wall. This arrangement was made with the view to possible future additions.

Plant motors of 50 hp. and less operate at 550 volts, three phase, while larger motors operate at 2200 volts. Squirrel-cage motors under 50 hp. are started by application of full voltage.

Electric power is carried to distribution centers in the plant through lead-covered cable in iron conduit. Power for lighting is distributed by a

2300-volt, three-phase cable feeding lighting transformers located near the load centers. An emergency lighting system fed by the power-house storage battery is arranged to go into service automatically in the event of failure of the alternating current supply.

Steam and Air

The plant steam consumption varies from 13 to 20 million pounds per month, depending upon the outside temperature. Most of this steam is produced by the anode and wire-bar waste-heat boilers referred to above. In addition to the waste-heat boilers, a third boiler is provided, which acts as a stand-by and carries the load when the other boilers are not in operation. It consists of a direct-fired, cross-drum boiler having a normal capacity of 500 boiler hp. and equipped with superheater.

All three boilers are on the same floor level and are operated and maintained by the Power Department. Water for boiler feed is taken from the river main and treated in a hot-process lime and soda water softener having a capacity of 10,000 gal. of water per hour. Feed water is delivered to the boilers by three duplex steam feed pumps.

High-pressure (100 lb. per sq. in.) compressed air is supplied to the entire plant by two 17 by 10½ by 12-in. motor-driven air compressors having capacities to deliver 750 cu. ft. per minute each. One unit acts as a stand-by but both can be operated at once. Low-pressure (30 lb. per sq. in.) compressed air is supplied by two 20 by 12-in. motor-driven compressors built to deliver 860 cu. ft. per minute each. One of these machines acts as a spare also.

Miscellaneous Data

A well equipped machine shop in the auxiliary building near the center of the plant takes care of all ordinary maintenance and repair work. A complete chemical and assay laboratory in the main office building handles all plant and control analyses.

Initial plant construction work was started in May 1930. The first wire bars were shipped in May 1931. During the 29 months that the plant has been in operation 126,333 tons of copper have been shipped to domestic and foreign markets and the "CCR" brand of wire bars has been favorably received in all these markets.

Electrolytic Copper Refinery of Ontario Refining Company, Limited, at Copper Cliff, Ontario

By Frederic Benard,* Copper Cliff, Ontario

(New York Meeting, February, 1934)

The electrolytic copper refinery at Copper Cliff, Ont., is operated by the Ontario Refining Co., Ltd. The associated companies behind the enterprise are the International Nickel Co. of Canada, Ltd.; The American Metal Co.; The Consolidated Mining & Smelting Co. of Canada, Ltd.; and Ventures, Ltd. The plant was the outgrowth of the rapid development of copper resources in the nickel-copper ores of the Frood and other great mines in the Sudbury district, and of the desire to have all primary production units grouped together as closely as possible.

The refinery is near the intersection of the Sault Ste. Marie to Sudbury highway and the Canadian Pacific Railway. It is about one mile south of The International Nickel Company's smelter at Copper Cliff. The standard-gage track systems of the refinery and of the smelter are connected by a standard-gage timber railroad trestle that crosses the railway and the highway by means of steel girder spans.

The Canadian National Railway has access to the plant by means of this trestle and the nickel company's tracks, so that the refinery has connections with both the Canadian Pacific and the Canadian National railways, as well as a direct connection with The International Nickel Company's smelter, from which the bulk of the copper to be refined is hauled by the latter's electric locomotives.

Buildings

The refinery (Fig. 1) covers approximately 60 acres, enclosed by an unscalable chain-link fence. The main buildings have a content of 23,000,000 cu. ft., with a total roof area of approximately 600,000 sq. ft. and near them are the substation, cooling pond and main standard-gage track system (Fig. 2). A ladder track for car storage extends 1200 ft. south of the coal-pulverizer tracks and the scale house is on this spur. The intake and pumphouse for fresh water supply are at Kelly Lake, about 3000 ft. south of the plant.

Owing to the severity of the winters, when temperatures of from 20° to 40° below zero are to be expected, the building of this refinery involved a number of problems in insulation, layout, ventilation, etc., which were

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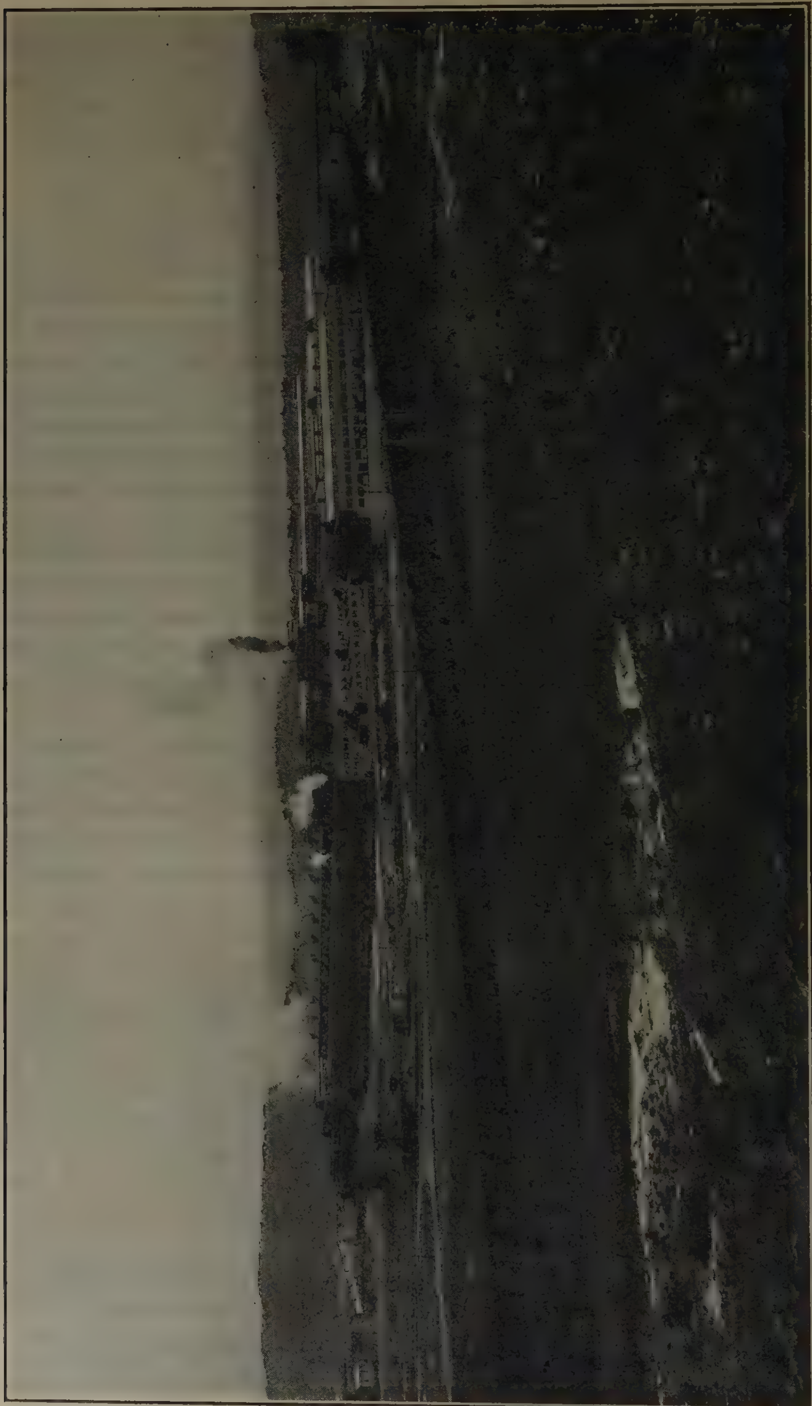

FIG. 1.—THE ONTARIO REFINERY AT COPPER CLIFF.

rather unusual in such construction. All the buildings except storage sheds and acid plant are of the same general type; as follows:

Steel frame, 8 or 12 in. brick and 8 in. tile walls supported on concrete foundation walls and footings. The concrete foundations are carried

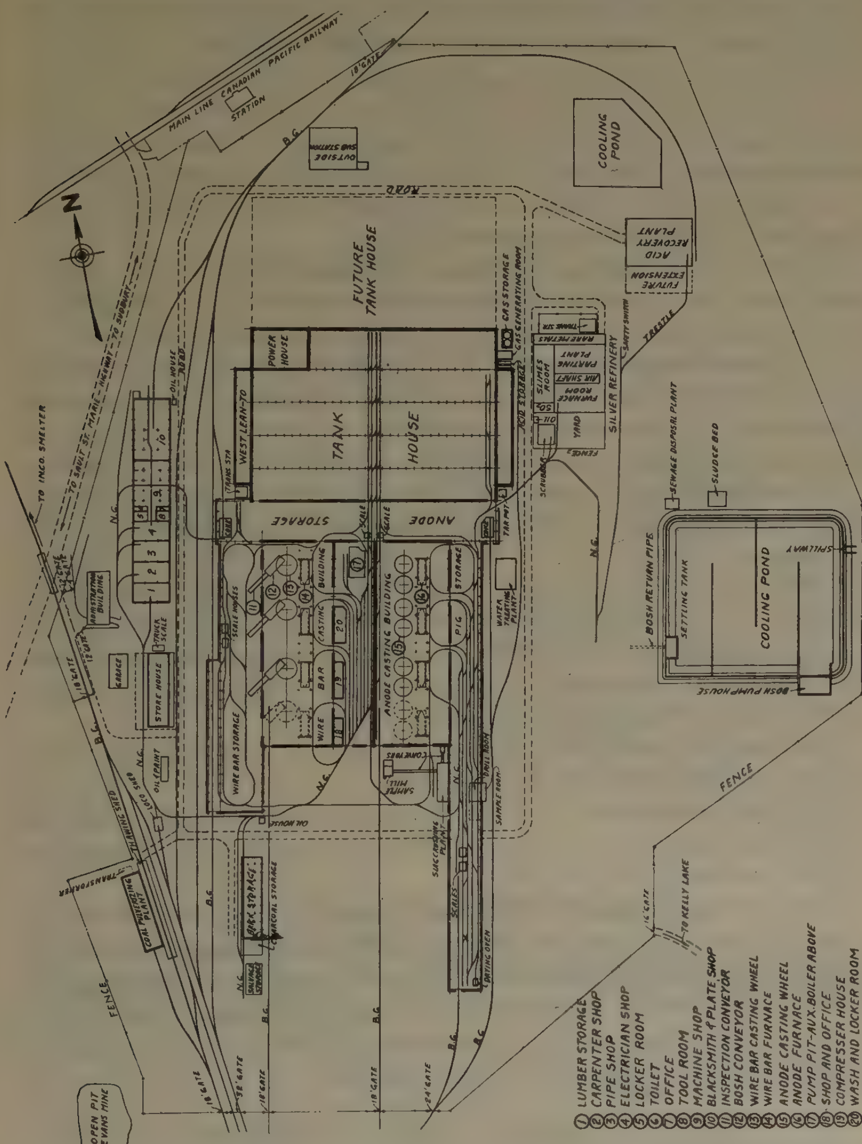

FIG. 2.—GENERAL PLAN OF ONTARIO REFINERY.

down 6 ft. below ground level and are brought to a height of 1 ft. above the ground. Roofs are poured-in-place gypsum, supported on steel trusses with a specification roof covering applied to gypsum slab. All buildings have a 6-in. concrete floor on slag or gravel fill.

Storage sheds have concrete foundation walls and footings with 8-in. brick or 10-in. concrete walls carried up 4 or 8 ft. above ground level, and transite siding from the top of brick or concrete to the roof.

The acid plant has 8-in. brick walls supported on concrete foundations with a 2 by 4-in. laminated wood roof laid on timber trusses carried on timber columns. The ground floor is concrete, 6 in. thick.

Capacity

The refinery has a present production capacity of 20,000,000 lb. of electrolytic copper per month. It consists of the following principal units:

1. Tank house for 1216 concrete regular depositing tanks with a capacity of 20,000,000 lb. per month.

2. Casting and storage buildings for 40,000,000 lb. per month. Casting-building equipment is installed with a capacity of 30,000,000 lb., including anode and wire-bar furnaces, casting wheels, cranes, etc.

3. Silver-refinery building, for the refining of silver and gold and the recovery of platinum and palladium, as well as the separation of selenium and tellurium. The equipment is installed for the present production of 20,000,000 lb. of copper per month, with provision for future extension.

4. Acid plant, for the recovery of nickel residue and acid, to handle the segregated electrolyte from the 1216 regular depositing tanks in the tank house.

5. Bosh cooling-water system, including pump house, cooling pond, piping, etc., with a cooling capacity for 40,000,000 lb. of copper per month.

6. Coal-pulverizing plant with foundations for an additional drier and a third coal pulverizer, which would give a pulverizing capacity sufficient for 40,000,000 lb. of copper per month.

7. Shops, administration building, etc., most of which have facilities for production of 40,000,000 lb. per month.

The refinery has been in operation a little over three years, starting anode-furnace operations in June, 1930, and electrolytic refining on July 1 of that year.

After a record-breaking construction program, completed in only 15 months, the plant soon settled into its stride, and the usual starting-up difficulties were rapidly overcome. The rated capacity of the plant has never been reached, of course, owing to depressed business conditions. It may be said, however, that the present installation has proved its ability to produce in excess of its rated capacity.

Weighing, Checking, Drilling and Sampling

Incoming blister copper is received in box cars which are sent to the pig-storage building. This building is 880 ft. long, 60 ft. wide, and 40 ft. high and contains two 10-ton overhead cranes. The pig is unloaded by two gantry cranes on to double-truck narrow-gage cars. It is weighed

and check-weighed on twin 20-ton track scales checking to 1 lb. All weighing and checking, drilling and sampling operations are carried on in the presence of shippers' representatives. A certain proportion of the pig, depending on the values contained, is sent for sampling.

A $1\frac{7}{32}$-in. hole is drilled in each pig by the continuous template method, with an electrically driven radial drill. Drillings from the pig are ground, riffled and screened according to standard methods, samples being taken for the plant laboratory, the shipper, umpire and reserve.

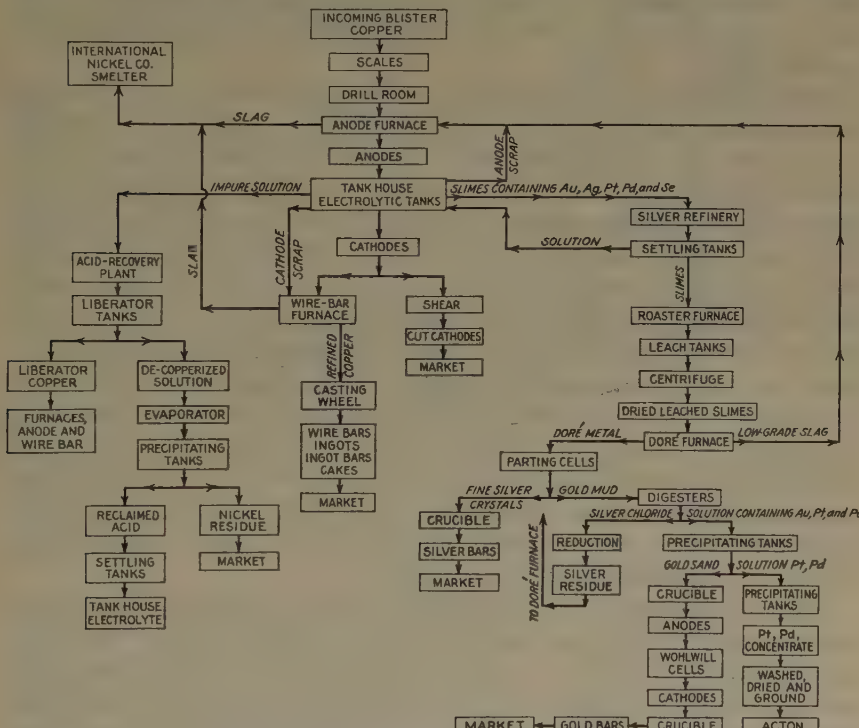

FIG. 3.—FLOW SHEET, ONTARIO REFINING CO.

Moisture determinations are made in an electric oven, automatically controlled by a thermostat. For drying blister, the temperature is held within a range of 300° to 350° F.

Sampling of slags from the anode and wire-bar furnaces is carried out in a mill equipped with grizzlies, a 12 by 20-in. primary jaw crusher, a cone crusher, mechanical conveyors, trommels, three Vezin samplers and two sets of rolls. The cone crusher and trommels permit the recovery of 20 to 25 per cent of the total metallics in the slag to be returned to the furnace. The crushed samples are melted in a graphite crucible in a small oil-burning furnace, and the resulting bar and slag are drilled and ground, final samples being sent for assay.

Anode Furnaces and Storage

This building is 403 ft. long and 149 ft. wide, divided into three sections: a charging aisle 44 ft. wide, a furnace aisle 22 ft. wide and a casting aisle 83 ft. wide.

The charging aisle has two 4-ton overhead charging cranes, with a charging rate of 200 tons per hour. Above these are two 10-ton overhead service cranes, one of which has an auxiliary 3-ton hook and magnet for use in connection with a 1½-ton cast-iron weight for preliminary slag crushing in a pit at the south end of this aisle and adjacent to the slag-crushing plant.

There are three reverberatory anode furnaces and space for a fourth. These furnaces are rated at 300 tons capacity, but charges up to 350 tons are handled in regular practice. The inverted arch of the furnace bottom is of concrete, which rests on cast-iron plates supported by 4-ft. concrete piers. The furnace bottom itself consists of two 12-in. layers of silica brick. The walls are 15 in. of magnesite brick below, and 15 in. of fireclay brick above the metal line, while the roof is of 20-in. silica brick. The verb is of chrome brick, and chrome dividers are used throughout the furnace between silica and magnesite. Buckstays are protected by copper castings, machined and drilled for circulation of cooling water, with provision on the furnace side of the casting to permit replacement of clay brick. Water-cooled steel skewbacks are used. Four charging bays are in each furnace, all 6 ft. 4 in. wide by 3 ft. 8 in., 3 ft. 4 in., 3 ft. 1 in. and 2 ft. 9 in. high, respectively. The first two bays have water-cooled doors; the other two have doors of clay brick in a steel frame. All doors are operated by hydraulic lifts.

Furnace operations have been intermittent, therefore no fair figure can be given as to number of charges per roof, walls or bottom. However, from experience to date, a roof should last 100 to 125 charges; clay-brick walls approximately 100 charges; magnesite walls below metal line about 200 charges; and bottoms 3 to 5 years, by patching top layer when needed.

Pulverized coal is used for fuel. Primary air is supplied by a 1460 r.p.m. fan delivering 3500 cu. ft. per min. at 8 in. static pressure, and secondary air is supplied by a 1440 r.p.m. fan delivering 13,500 cu. ft. per min. at 3 in. static pressure. Furnace draft is augmented by a fan 81 in. in diameter, which delivers 24,500 cu. ft. per min. at 4½ in. static pressure. The stacks are 4 ft. in diameter and 35 ft. above top of fan. Each furnace has three Warford burners. The firebox is 8 ft. 6 in. long and 13 ft. wide by 4 ft. high at the bridge wall. The hot furnace gases are utilized in waste-heat boilers placed above the furnaces.

Under the furnaces are inclined pits which slope towards runout pits under the charging floor. Any copper that runs into these pits is sectionalized and thereby rendered easy of removal by crane. Naturally,

the pits must be kept dry; they are equipped with automatic sump pumps and electric water alarms.

Material for furnace feed is brought in from the pig-storage building on convenient narrow-gage loops, which pass before the charging bays of each furnace. The first part of the furnace charge consists of a layer of "anode scrap" from the tank house, spread over the furnace floor to protect it from the impact of the copper pigs, which are charged in next. An average of 325 tons is charged in less than 2 hr. About 25 tons of anode scrap is used later as a hot charge. Other miscellaneous feed to the anode furnaces includes silver-refinery slags and byproducts, tank-house and storage-building sweeps, bosh scale, metallics from slags, etc.

The furnace cycle is fairly constant at 22 hr. The melting period requires between 11 and 12 hr. Oxidizing, poling, casting and charging require from 8½ to 10½ hr. This cycle naturally varies with different types of charges. The furnace is charged directly after casting and is melted flat in 6 or 7 hr. In about 5 or 6 hr. more, the full charge is afloat and the oxidizing period begins. Six or eight iron pipes are inserted through portholes in the doors, and 20-lb. air is introduced into the bath. By this means, most of the easily oxidized impurities are slagged off. The oxidizing, or "flapping," period is continued until the oxygen content of the bath is over 0.60 per cent, which is determined by sampling in a "say" ladle at the skim door of the furnace. At this point the bath is covered with coke, and the reducing period begins. Large, green, hardwood poles are inserted through the skim door and forced under the bath by a 2-ton air lift. A charge of this size requires 5 to 6 tons of poles (mostly white and

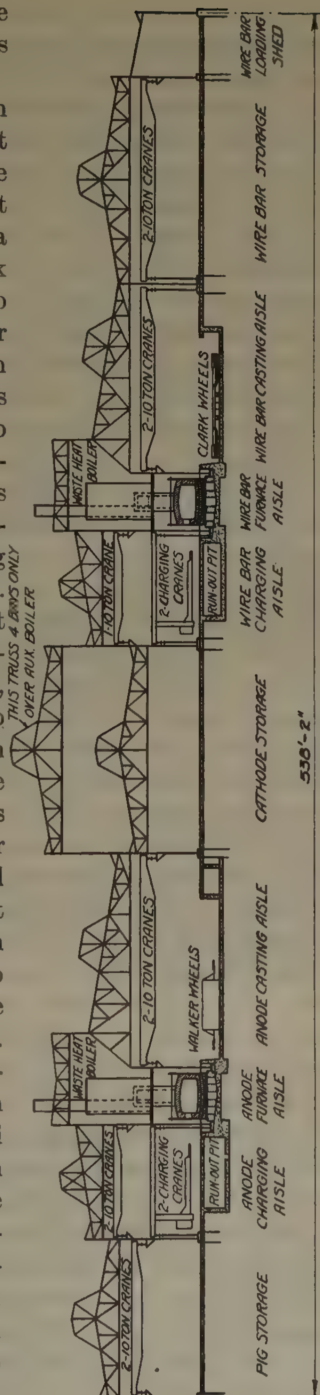

FIG. 4.—SECTION THROUGH CASTING BUILDING.

yellow birch). Poling continues until most of the oxygen has been removed. When the oxygen is sufficiently reduced, the set of the sample taken is flat, and the copper is ready to cast. The anodes produced run over 99 per cent copper. Their chief impurity is nickel, which averages about 0.45 per cent. The gold and silver vary according to the charge. The slag produced, averaging about 2 per cent of the furnace input, is crushed, sampled, and returned to the International Nickel Co. smelter. Coal consumption is 11 to 12 per cent of total charge.

A feature of each furnace is a double tap-hole. For anode casting there are two 36-ft. Walker wheels per furnace, each of which carries 22 (heavy-type) copper molds. The wheels are cable-driven, with an oil drive between the motor and the cable winch, giving perfect smoothness of operation. Anodes are cast at the rate of 85 to 100 tons per hour. When operating two furnaces the same casting crew can cast both furnaces within the 8-hr. day. As the wheel revolves between the pouring and take-off points, the anodes are cooled by sprays of water, first on the bottom of the mold and then on top of the anode. As the mold approaches the take-off point, the anode is partly lifted from it by a push-up pin beneath, hydraulically operated. The partly cooled anodes are then lifted by a one-man pneumatic take-off and placed in bosh tanks, the water from which circulates through the large cooling pond outside the building. These tanks hold 60 anodes. As the anodes are put down, the chain conveyors move forward 3½ in., coming to rest in the correct space for the next anode. Thirty-eight anodes at a time are picked up by a swinging rack on a 10-ton overhead crane, which places them in a rack car for weighing and transfer to tank house. The spacing of the hooks on the crane racks and of the notches on the rack cars is 3½ in. from center to center, conforming to that used in the tanks, which makes unnecessary any respacing of the anodes.

This building, 544 ft. long by 80 ft. wide, runs parallel to the tank house at right angles to the casting building. While primarily for the storage of anodes, it is large enough for the storage of cathodes and miscellaneous materials. It has two 10-ton overhead cranes. It contains the anode and cathode scale houses, the cathode shear, and pits at either end for storage of anode and wire-bar coke. There are also steel racks for the storage of several hundred loads of anodes. Rack cars of anodes from the anode-casting aisle are first weighed on a 20-ton track scale, and the anodes are then either stored in racks or run over a straight line of track into the tank house.

Cathode Shear.—This machine is at the west end of the anode-storage building. The shear is of the gate type, having a width of 42 in., a capacity of ⅝-in. mild steel, and a weight of about 15 tons. Cathodes are placed in front of the shear on narrow-gage cars and are lifted on to

the shear by a 500-lb. electric hoist. After cutting, they are dropped on to an inclined apron conveyor, from which they are loaded directly into box cars, or, if further cuts are required, they are returned by gravity on a roller conveyor to the front of the shear. Cut cathodes are used for remelting in the brass trade, and as anodes in the electroplating trade.

Tank House

The tank house is a fireproof structure measuring 480 by 340 ft. in ground plan. A most particular interior paint job was done to give protection against the moist acid fumes. The gypsum roof and steel-work were given three coats of paint and finished in aluminum. The

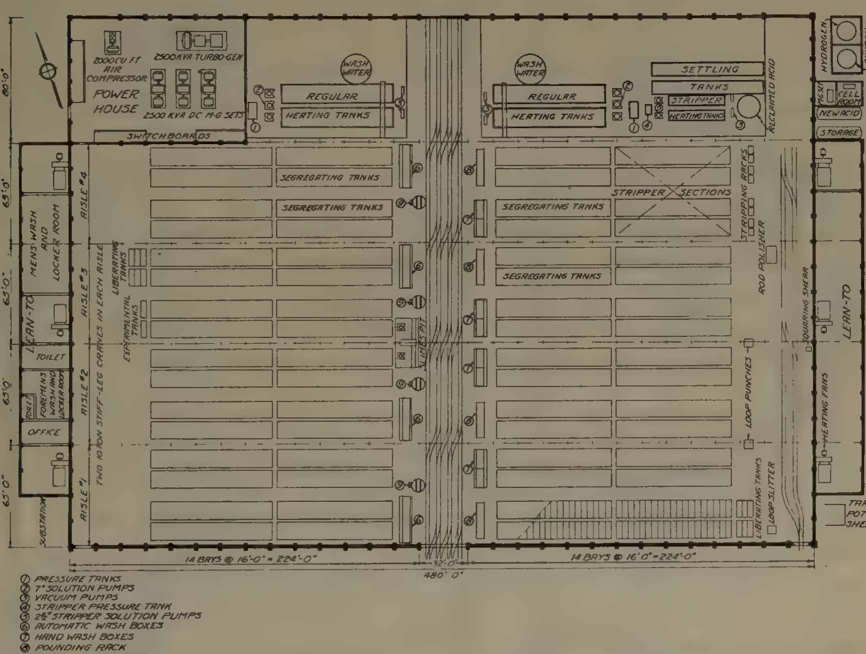

FIG. 5.—PLAN OF TANK HOUSE.

general effect is pleasing, and the light reflection is very good. The cellar has an unusual amount of headroom, the distance from the floor to the tank bottoms being 9½ ft. The cellar floor is of concrete overlaid with vitrified brick laid in asphalt. Asphalt is also used in the joints and is ironed over the surface of the brick, making a particularly smooth and acidproof surface. Mastic fillets from 3 to 5 in. high are laid at all walls, column footings, piers, etc., as protection against acid corrosion at the base. All parts exposed to drip are painted with black asphaltum. Whenever possible, pipes, launders, busbars, etc., are placed overhead.

The main working floor is also of concrete covered with vitrified brick laid in asphalt. The bricks have a very low absorption ratio, and on

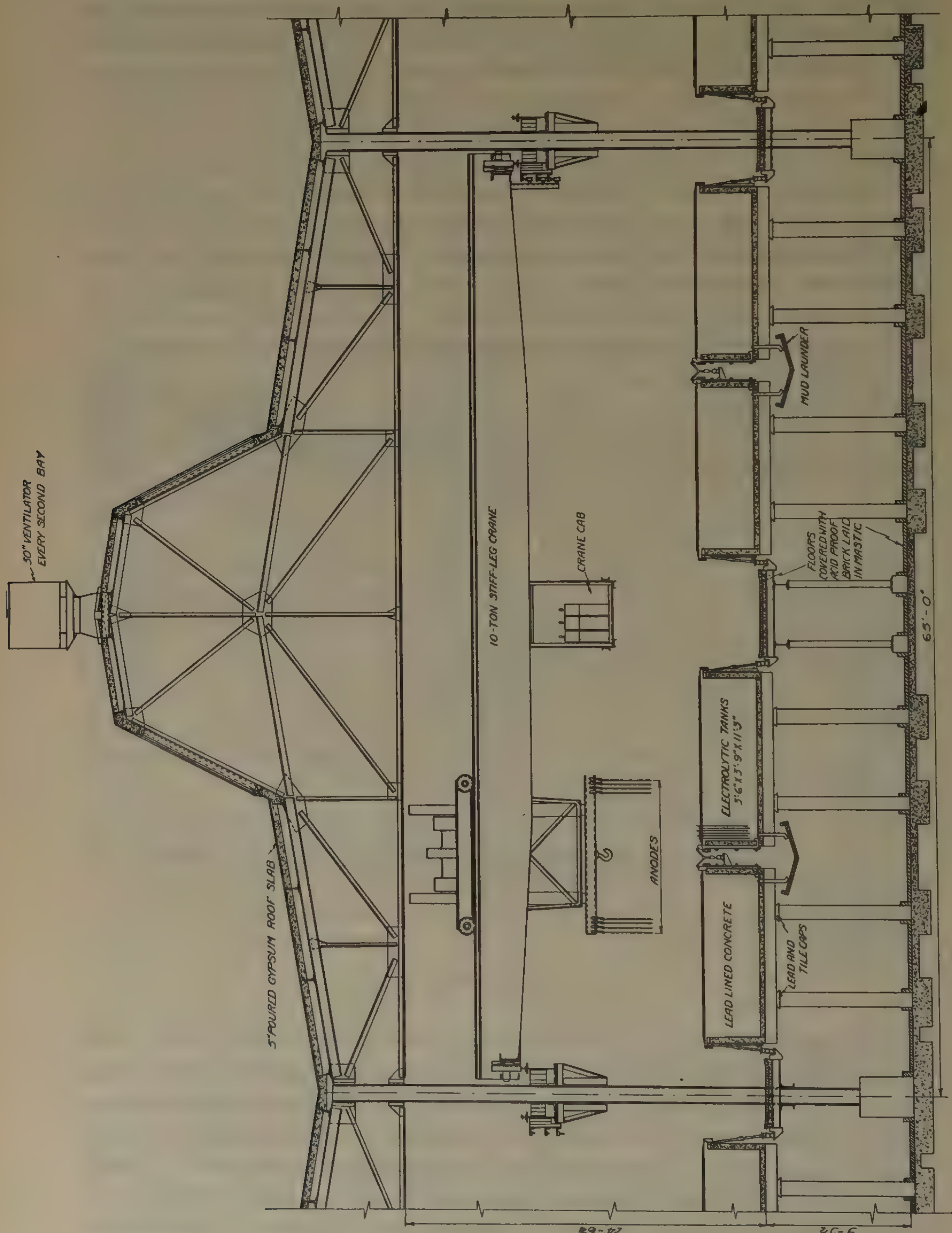

Fig. 6.—Cross-section through Tank House.

this floor they are not covered with asphalt. The asphalt joints between the bricks were made with great care. The rough surface of the bricks

gives a very satisfactory nonslip working floor, which is kept clean by the use of water and brooms. Some bricks taken up recently showed no penetration by acid either through the brick or joints.

Double windows are provided for good insulation, but most of them are hinged to give natural ventilation when desired. There is overhead escape for moisture-laden air through ventilators in the monitors and forced ventilation is provided when necessary by six large blower fans. Each fan is capable of delivering 35,000 cu. ft. of free air per minute. The air is delivered through large iron ducts to all parts of the main working floor and cellar, and can be heated by steam radiators in each fan. However, the hot solution in the tanks provides most of the heat necessary to keep this large building warm. The fans are used chiefly for bringing in fresh, dry air to keep down humidity and prevent condensation.

The tank house is arranged in four depositing aisles running east and west, with a parallel machinery bay on the north. This arrangement permits ready expansion by the addition of more depositing aisles to the north, leaving the machinery bay in the center of the system. Work aisles run north and south—a center aisle 64 ft. wide and an aisle 48 ft. wide at the east and west ends of the building. The center aisle gives a straight-line narrow-gage railway transport of anodes from the casting building into the tank house, and of cathodes from the tank house to the cathode-storage building. Within the tank house, all material is transported either by overhead cranes, of which there are nine 10-ton units, or by narrow-gage cars pulled by electric winches.

The tank house contains 1230 concrete tanks, lined with lead, laid out in 32 sections of 38 tanks per section, with two liberator units of six tanks each and two full-size experimental tanks. The tanks are constructed of precast, reinforced concrete slabs with interlocking ends. The concrete mixture used was 1:1½:2½ reinforced with two layers of ¼-in. welded wire mesh. The ends of the reinforcing mesh were well covered to prevent short circuits between tanks, bolts, etc. The sides of the tanks are 4¾ in. thick; the ends, 3½ in.; and the bottoms, 4 in. The tanks are 11 ft. 3 in. long, 3 ft. 6 in. wide, and 3 ft. 9½ in. deep, inside dimensions. The tanks are supported by concrete piers, with a 2-in. slab of vitrified tile and a lead cap between the pier and the tank sill; the tile being for electrical insulation from ground, and the lead cap for the protection of the pier from drip. The tank linings are of 6-lb., 6 per cent antimonial lead. The concrete bottom slabs are provided with a number of "weep holes" for escape of solution in case of leaks.

The experience with these concrete tanks to date has been very successful, justifying the departure from the older wooden construction. There has been no evidence of short circuits through the reinforcing, and, with proper care, a long life for the tanks is indicated.

The anodes consist of impure copper, partly refined in the anode furnace, and containing the precious metals that came in the original blister. They are fed to the tanks one full tank charge at a time, the 38 anodes in the load being suspended from a hook rack on a 10-ton stiff-leg crane. The anodes weigh about 530 lb. apiece. This weight is intended to leave about 13 per cent anode scrap at the end of a 28-day run. The anodes measure 36 by 36 in. by about 1¼ in. thick. They are suspended in the tanks by means of two lugs and are spaced 3½ in. from center to center. Between the anodes, starting sheets are hung by hand. These are thin sheets of refined copper suspended by two loops of the same material from removable copper cross rods. The cathodes are removed at the end of 14 days, two crops of cathodes being pulled for each charge of anodes. The crane lifts 38 cathodes from a tank in one load.

The electrolyte is primarily a solution of copper sulfate, with free sulfuric acid added to increase the conductivity. The copper content is held at about 3 per cent with 13 per cent free acid. By the electrochemical action, the anode copper is dissolved and redeposited on the cathode, the soluble impurities such as nickel, iron, arsenic and antimony being dissolved in the solution, and the insoluble ones, such as lead, selenium, tellurium and all the precious metals remaining on the surface of the anode or falling to the bottom of the tank as slimes. In order to promote the speed and efficiency of deposition, the electrolyte is heated and circulated through the tanks. Solution resistance constitutes about 50 per cent of the total resistance of the circuit, and, since the conductivity of the solution is a function of temperature as well as of free acid and various soluble impurities, an economic balance is struck between these various factors. The adopted practice is to heat the solution to 150° F. and circulate it through the tanks at from 2½ to 3 gal. per minute. The usual addition agents, glue, oil and bindarine, are added in small quantities to promote the toughness and smoothness of the cathode deposit. The solution enters one end of the tank at the bottom and overflows at the top of the other end. Individual circulation is provided for each tank through a ⅝-in. rubber hose inlet and ¾-in. outlet. The return solution flows through lead pipes to the heating tanks in the machinery bay. In these tanks, which are 72 ft. long, 11 ft. 6 in. wide, and 2 ft. 3 in. deep, the solution is heated by flowing counter-current to 10-lb. exhaust steam carried in forty 1½-in. lead pipes running the length of the tank. From these tanks the solution is lifted by vertical centrifugal lead pumps to head tanks, from which the solution flows out under a gravity head of about 6 ft. to the various sections.

With anodes at 99 per cent copper, the copper content of the solution tends to build up beyond the desired 3 per cent. To reduce it, the solution is passed through liberating tanks, using insoluble hard lead anodes and regular cathode starting sheets, the solution being depleted of copper by the amount of the cathode deposit in these tanks.

The depositing tanks are arranged according to the Walker multiple system, all tanks being in series and the 38 anodes and 39 cathodes in each tank being in parallel. A current of 10,500 amp. is passed through the tanks, the circuit voltage required varying from 60 to 160 volts, depending on the load. The voltage drop across each tank varies with conditions, usually 180 to 200 mv. There are two electrical circuits crossing at right angles two solution-circulation systems. By this means, the maximum potential difference between any two points on one circulation system is reduced to one-half the circuit voltage, and the loss through stray currents and copper growths in unwanted places, pipe lines, etc. is correspondingly retarded.

A system of tank inspection is maintained, in order to prevent short circuits between electrodes, to insure proper spacing and clean contacts, and to obtain the best and smoothest deposit at the lowest power cost. The actual production is divided by the theoretical production in pounds per ampere-day, and the ampere efficiency thus obtained usually ranges between 97 and 98 per cent.

The starting sheets for the cathodes are prepared in two special stripper sections charged with extra heavy anodes and rolled copper mother blanks. These sections have a separate circulation system. There are 31 anodes, 30 mother blanks, and two end sheets in each stripper tank. The sheets are stripped from the blanks with steel knives every 24 hr., the vertical system of stripping being employed. Each blank gives two starting sheets, one from either side, which weigh about 10 lb. each. Loops 4 in. wide of the same material are punched into them by electrically driven double-punch machines. After being straightened by hand, the sheets are ready to be charged to the regular depositing sections, from which they later emerge as cathodes weighing about 240 lb. each. The removal of the thin sheet of copper from the mother blank is facilitated by the use of a thin film of oil on the face of the blank. The sides and bottom of the blank are grooved, giving a clean break of the sheet along these edges.

For the removal of impurities, a given amount of electrolyte must be removed from the tank-house circulation each day. The principal solution impurity is nickel, which markedly increases the resistance of the solution. When removing this nickel, the other impurities are easily kept below the desired limits. The first step is to pass the solution through the Pyne-Green segregating tanks, which have a bottom inflow, a bottom outflow, and a small orifice near the top through which passes less than 10 per cent of the total flow. By means of the restricted circulation at the top, a segregation takes place and the upper layers of solution are depleted of their copper ions, the copper content being reduced from 3 to 1 per cent, while the nickel and acid content is somewhat increased. This "segregated" solution is then delivered by automatic

airlifts to the acid recovery plant (referred to later). The importance of this step can be judged by the fact that 10 times as much power is required to liberate copper from solution by the use of insoluble anodes as is required in regular depositing tanks using copper anodes.

The electrolytic slimes containing the precious metals are cleaned from the tanks every 28 days. They pass through a launder system to a central collecting tank and are delivered by 2½-in. vertical hard-lead pumps to the silver refinery (referred to later).

The cathodes, after being pulled from the tanks, are first washed with hot water under 120 lb. pressure to remove solution and traces of bluestone. They are then piled on to narrow-gage cars, weighed on the cathode scale, and delivered to the cathode-storage building, situated between the anode and wire-bar furnace departments, ready for delivery either to the cathode shear or to the wire-bar furnaces.

Wire-bar Furnaces and Storage

The charging aisle, furnaces and waste-heat boilers in the wire-bar furnace building are practically duplicates of the anode-furnace equipment; the casting aisle, however, is 73 ft. wide. These furnaces also are rated at 300 tons capacity. Because of the longer cycle, they are not charged as heavily as the anode furnace. About 320 tons constitutes a normal charge. Cathodes required for the market in sheared form reduce the wire-bar furnace operations to less than those of the anode furnace.

The wire-bar furnaces operate on the same principle as the anode furnaces, except, of course, that the time is longer for refining and casting. The furnace cycle is fairly constant at 24 hr., as follows:

	Hours		Hours, Approximately
Charging	1½ to 2		
Melting period	10 to 11	Flat	6½
		Afloat	4
Oxidizing, poling and casting	11 to 12	Flapping	4
		Poling	3
		Casting (averaging all sizes cast)	5

The oxidizing period is carried on by air pipes carrying 20-lb. air, introduced at the door ports and two extra ports cut in the back wall of the furnace. A rabble is used for flapping at the skim door. Say-ladle samples are taken to indicate the progress of refining. When an optical oxygen content of 0.90 per cent is obtained, blowing and flapping is stopped. Care is taken not to go much higher than 0.90 per cent oxygen, because the copper itself will oxidize readily above that point. For

furnace control the approximate oxygen content is determined by microscopic examination of the samples taken during this period. It is later determined by chemical means. Slag is removed twice during this period. Slag, minus metallics, is approximately 1.75 per cent of the total charge.

When the oxygen content is 0.90 per cent and the bath has been skimmed clean, a covering of low-sulfur coke is shoveled in the skim door to keep out impurities from the burning coal and prevent subsequent reabsorption of oxygen during the poling period. The copper is now brought up to the proper pitch by poling up with green poles (pole consumption is 8 to 9 tons per charge). The tough-pitch point is determined by taking block samples and watching the set. When ready to cast the oxygen content is about 0.028 per cent (optical).

Typical temperature range of the copper during the melting, flapping and poling periods ascertained by optical pyrometers is as follows:

	Deg. F.
"Off bottom" or afloat	2150
Flapping early stages	2140
Flapping before coking	2175
Poling about 1 hr. after coking	2125
Poling about 2 hr. after coking	2100
Ready to cast about 3 hr. after coking	2070
Casting	2060

Coal consumption is 12 to 13 per cent of the total charge.

The poled-up copper is now ready for casting into whatever refined shapes may be required, such as wire bars, ingots, ingot bars, cakes or transferred to holding furnaces for casting billets. The 40-ft. Clark casting wheels used are carefully built to secure stiffness and smooth running. The wheels are cable-driven with an oil drive between the motor and the cable winch, permitting smooth acceleration and deceleration and resulting in a uniform product free from edges, etc. The wheel accommodates 30 (four-depression) wire-bar molds and 20 ingot or ingot-bar molds. Two ladles are used, casting into two molds at a time. The molds are cooled by water sprays from beneath. The copper shapes drop on to a conveyor placed at an angle of 45° from the building, which takes them out of the bosh water and delivers them to an inspection conveyor 35 ft. long in the wire-bar storage building.

Very rigid standards and specifications have been laid down and adhered to. All castings are inspected accordingly and any that fail to conform to these standards are returned to the furnace. Cracked wire bars are not encountered for the reason that all molds are kept in first-class condition, their temperature maintained at a uniform degree during casting, and the copper entering the molds is carefully controlled. A uniform fine "crown set" is maintained on all bars. An average of

97 per cent good production, exclusive of warmers, is obtained under present practice.

The Ontario Refining Co., Ltd., is licensed to cast cakes by a patented process. Various sizes of cakes, from 185 to 600 lb. in weight, are cast. These "ORC" V. C. cakes are parallel-sided and are vertically cast in water-cooled molds. The special manner in which V. C. cakes are produced offer the manufacturer of sheet copper many advantages.

Billets (electro and phosphorized) are cast in 60-in. copper water-cooled molds located on a small-diameter wheel at the south end of the casting aisle. For this purpose, 5-ton holding furnaces are used. These small furnaces receive their charge of copper from the second tap-hole of the large furnace and are transported by one of the overhead service cranes to an electrical tilting rack adjacent to the small wheel. The second tap-hole on the large furnace is also used for obtaining copper for mold making. For wire-bar molds, the copper runs through a launder directly to a near-by mold press which is equipped with machined-copper water-cooled cores for making the required depressions. These copper water-cooled cores have proved very satisfactory, but their length of service is not yet determined. A bull ladle carried on the overhead crane is used when anode molds are cast in the foundry at the extreme south end of the casting aisle. The anode molds are cast in sand mother molds that have been thoroughly dried in an oil-fired oven.

ORC copper fully meets specifications 198 and 202 of the British Engineering Standards Association as well as specifications of the American Society for Testing Materials. A typical analysis of ORC copper is given in Table 1.

Table 1.—*Typical Analysis of ORC Copper*

	Per Cent		Per Cent
Antimony	0.0010	Oxygen	0.0300
Arsenic	0.0005	Selenium	0.0018
Bismuth	0.0000	Silver	0.0012
Cobalt	Trace	Sulfur	0.0018
Iron	0.0013	Tellurium	0.0014
Lead	0.0003	Tin	0.0000
Manganese	0.0000	Zinc	0.0000
Nickel	0.0016	Copper, excluding silver	99.9600
			100.0009

The wire-bar storage building, 532 ft. long and 80 ft. wide, runs parallel to the wire-bar casting building, with a loading shed at one end 293 ft. long and 25 ft. wide. Its equipment includes both broad-gage and narrow-gage trackage, two 10-ton overhead cranes, twin scales, and gantry cranes for the loading of refined shapes. From the inspection conveyors, which convey the copper from the casting aisle to this build-

ing, the refined shapes are taken by air lift, stocked on cars and sent to scales. After weighing, they are either loaded for shipment or stored in stock piles. Thus the handling of the copper is almost entirely mechan-

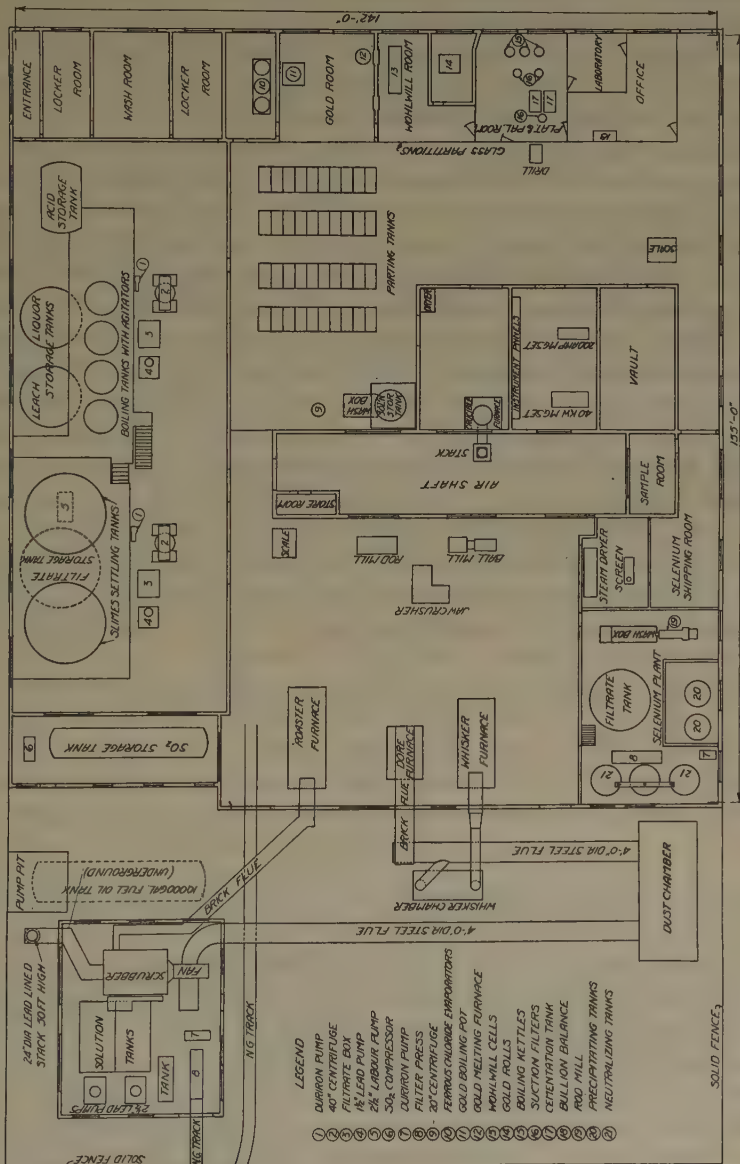

FIG. 7.—PLAN OF SILVER REFINERY.

ical and the method of passing it from one building to the other via the inspection conveyor eliminates narrow-gage transportation between the two buildings.

Silver Refinery

The silver refinery occupies a fenced-in area 243 ft. long and 142 ft. wide, adjacent to the northeast corner of the tank house. The main building, 155 ft. long and 142 ft. wide, is in the form of a U, with the slimes room and parting plant at right angles. In addition to these three main divisions, there are also the flue products and scrubber plant; the gold, platinum and palladium, and selenium departments.

The slimes from the tank house, along with tank-house electrolyte, are received in two lead-lined wooden tanks 16 ft. in diameter by 6 ft. deep, on an upper floor in the slimes room, with a third tank of the same size and construction directly underneath on the ground floor. The two upper tanks are connected by a lead-lined launder and the clear solution can flow from one to the other and then overflow into the lower tank. All tank outlets are shrouded so that any float slimes are retained.

Slimes are allowed to settle 8 to 10 hr. in the upper tanks before the clear solution is siphoned off into the lower tank. After further settling, this clear solution is pumped back to the tank house by a 2-in. lead pump. From the settling tanks the slimes are fed through two discharge pipes into a 40-in. suspended centrifugal equipped with a rubber-covered steel basket. The liquid-solid ratio is about 1 to 1, and the overflow is regulated by means of a 1½-in. duriron plug cock. A centrifuge charge consists of approximately 700 lb., and the dried product contains about 20 per cent moisture. A typical analysis of these slimes is given in Table 2.

Table 2.—*Typical Analysis of Slimes*[a]

	Per Cent		Per Cent
Copper	24.70	Silica	0.18
Nickel	19.80	Lead	1.51
Selenium	15.03	Arsenic	0.24
Tellurium	3.61	Antimony	0.32
Iron	0.40	SO_4	5.48

[a] Gold and silver content of slimes has purposely been omitted because of the wide variation of these metals in custom materials treated.

In order to remove the copper and the nickel, the slimes are given a sulfatizing roast. Just enough concentrated sulfuric acid is mixed with the raw slimes to combine with the copper and the nickel. This sulfatizing roast is carried out in a reverberatory furnace with a sectional cast-iron hearth 18 ft. long and 6 ft. wide. The furnace is oil-fired and is equipped with two burners, one at the end of the furnace and the other halfway down the side. A charge consists of 1700 lb. of raw slimes and requires 6 hr. for complete sulfatizing. The charge is continually rabbled during the sulfatizing period.

The roasted slimes are raked from the furnace into round dust-tight cannisters, which have a conical top, and are transferred in buggies to the

slimes room for leaching. Each cannister holds about 375 lb. of roasted slimes.

The roasted slimes are leached in two lead tanks, 6 ft. in diameter and 7 ft. deep, on an upper floor. The tanks have a skeleton steel support and discharge plugs in the bottom. Agitation is obtained by means of a heavy Ascoloy-covered lead impeller driven through bevel gears from a transmission shaft that runs over the center of the tanks. The mechanical agitation is augmented by a ¾-in. lead pipe extending to the bottom of the tank, through which 40-lb. air is supplied. The tanks are also equipped with a steam line for heating the solution.

The roasted slimes are hoisted up to the leaching tanks. A charge consists of approximately 5400 lb. It is given one leach with 10 per cent sulfuric acid and one water leach. The solution is brought to almost the boiling point and is agitated for one hour. After settling, the leach liquor, consisting mostly of copper sulfate and nickel sulfate, is decanted off. After the water leach, the residual pulp is discharged into the receiving tank of No. 2 centrifuge. This machine and auxiliary equipment are similar to the raw slimes centrifuge. After washing thoroughly with hot water, the leached slimes are discharged into steel boxes and transferred to the furnace room for treatment in the doré furnace.

The leach liquor from the roasted slimes is pumped to one of two cementation tanks, which adjoin the leaching tanks and are of similar design. Here, any selenium, tellurium, or silver present in the leach liquor is cemented out with copper sludge. After this cementation treatment the liquor is pumped to the tank house by means of a 1½-in. vertical lead pump. The slimes in the cementation tanks are centrifuged and given the same treatment as the raw slimes.

The treated slimes are smelted and refined to doré metal in an oil-fired reverberatory furnace. This so-called doré furnace measures 8 by 5 ft., inside dimensions, with a height of 3 ft. 4½ in. at the firebox and 2 ft. 3 in. at the uptake. The bottom is supported by a 1⅛-in. cast-iron plate which rests upon concrete pillars. Originally, the bottom was constructed of one course of 9-in. magnesite laid on top of a concrete slab in the form of an inverted arch and having a 4-in. spring. When the furnace was rebuilt after being in operation for two years, a monolithic bottom 6½ in. thick was installed. The side walls are of 9-in. magnesite brick. The roof is built of 9-in. clay brick. The furnace is heavily reinforced with 1⅛ in. cast-iron plates along both sides and the ends. Buckstays constructed of two 7-in. channels placed back to back resist the thrust of the hearth and roof. The buckstays are tied together top and bottom with 1-in. rods.

A doré furnace charge consists of approximately 14,000 lb. of treated slimes containing 20 per cent moisture. The charge is introduced into the furnace in batches of 1000 lb. at a time and each batch is melted down

before the next one is added. The slimes are mixed with a flux of 1 per cent sand and 2 per cent fluorspar, and slag is skimmed before the furnace is fully charged. After the charge is melted and the scoria slag skimmed, the matte is refined to doré with soda ash and niter. A treatment of a charge requires approximately 48 hours.

The scoria slag is crushed and returned to the anode furnaces. The soda-niter slag is leached with water, the residue is charged into the doré furnace and the leach liquor is pumped to the selenium plant.

The doré is hand-ladled from the furnace and cast into plates in water-cooled copper molds. These plates are approximately 17 by 9½ by ½ in. in size, and form the anodes in the subsequent electrolytic treatment in the parting plant.

The parting plant with its attendant operations extends the full length of the north side of the building. Along the north wall are the office and weighing room, the platinum-palladium room, Wohlwill room, and gold-melting room; all partitioned with glass from the parting plant.

The parting process is carried out in 40 Balbach cells laid out in four rows of ten each. Each row of cells rests upon two 8-in. I-beams and separated from them by 3½-in. concrete blocks. The I-beams are supported by concrete piers. The floor underneath the cells has a slope of 1½ in. to the foot, so that any spillage or leaks will run into a mastic-lined gutter.

The cells are concrete lined with mastic and measure 52 by 24 by 9 in., inside dimensions. Carbon and graphite plates ½ in. thick are fitted into the bottom of the cell and form the cathode.

The basket holder is constructed of long-leaf yellow pine, and measures 31 by 24 by 8 in. inside dimensions. Glass rods ¾ in. in diameter, resting on a grooved shoulder on the inside lower ends of the basket holder, support the basket. These baskets consist of a rectangular framework made from yellow pine, measuring 29½ by 20½ by 5½ in. inside dimensions, around which is placed a snugly fitting canvas bag made from No. 8 duck. The interior of the basket is lined with a piece of muslin which is roughly shaped to the basket. The edges and ends of the muslin are tucked into wooden rods along the sides.

A charge to a cell consists of five doré plates weighing about 70 lb. The cells are connected in series, and connections from the overhead busbars are made through flexible copper cables to a silver "candlestick" contact. The cells operate at 185 amp. The average voltage drop across each cell is 4 volts. Current is supplied by a motor-generator set with a range of 40 to 200 volts and 150 to 200 amperes.

The doré anodes average 992 fineness, which is chiefly silver, but contains appreciable quantities of gold, platinum and palladium. The electrolyte is a solution of silver nitrate and copper nitrate containing

50 grams of silver and 75 grams of copper per liter and 1 gram per liter of free nitric acid.

The cells are scraped every 4 hr., and every 8 hr. the silver crystals are removed from the cell by a perforated aluminum shovel. After draining, the crystals are transferred to a stoneware tank, equipped with a perforated false bottom covered with muslin, which acts as a filter. The tank is protected by a wooden framework and is mounted on wheels. It will hold 9000 oz. of crystals. When full, the crystals are thoroughly washed with hot, distilled water to remove the copper. After draining, the crystals are transferred to the melting room, where they are melted in a graphite crucible in an oil-fired furnace and cast into bars weighing 1000 troy ounces.

Gold slimes are collected every 48 hr. The cell is short-circuited and any pieces of unparted doré are washed free of slimes and transferred to an adjoining basket. The inner muslin lining containing the slime is gathered together and the solution squeezed out. All soluble silver is removed in a 20-in. rubber-lined centrifuge of the supported type. Five bags of slimes at a time are evenly distributed in the machine and centrifuged until all solution has been removed. The charge is washed with hot water and dried in the centrifuge. Slimes are scraped from the bags into a steel box and transferred to the platinum-palladium department.

The platinum-metals department consists of three steam-jacketed, round, stoneware kettles, 50-gal. capacity, located on an upper level. Each kettle is provided with a treated concrete hood which in turn is connected to a 6-in. stoneware flue that leads to a standard stoneware tower through which a 15 per cent solution of sodium carbonate is continually circulated. A 6-in. stoneware exhaust fan draws the gases through the tower. One kettle is used for dissolving the slimes; the adjoining two for the precipitation of the gold.

In operation, a charge of approximately 125 lb. of slimes is introduced into the kettle and digested with aqua regia for 12 hr. After settling, the solution is decanted through a stoneware suction filter on a lower level. The filter medium is an asbestos pad covered with a filter paper on which is superimposed a layer of twill. Any silver present in the slimes is caught on the filter as a chloride. It is reduced with zinc and sulfuric acid, and returned to the doré furnace.

The filtrate containing the gold, palladium and platinum in solution is elevated through a Mariotte bottle into the precipitating kettles. Suction is provided by a vacuum pump with a capacity of 8 cu. ft. at 20-in. vacuum.

The gold is precipitated from solution by the addition of ferrous chloride. When precipitation is complete, the solution is allowed to settle and is decanted through a filter similar to the one used in the

previous step. The gold sand is washed thoroughly in the filter, dried, charged into a crucible, melted and cast into anodes for treatment in the Wohlwill cells.

The Wohlwill cells are of white porcelain, 10½ by 10½ by 12 in. deep inside, arranged in a battery of six cells, and are heated by means of hot plates, one plate to each cell. Electrode supports are made of silver. The anodes are cast with a hole in the top and suspended from the anode supports by silver hooks. The anodes measure 9 by 3 in. by ⅜ in. thick. The cathode starting sheets are strips of fine gold rolled out to a thickness of 0.005 in. and cut to proper length. One end is turned and punched to form a loop by which to hang it from the cathode rod.

In each cell there are two anode and three cathode rods and from each rod are suspended three cathodes or three anodes. Direct current is used, supplied by a motor-generator set of 200 amp. capacity and 1 to 12 volts. The cells operate at 175 amperes.

The electrolyte is a solution of gold chloride containing 90 to 100 grams of gold and 100 grams of free hydrochloric acid per liter. Agitation of the electrolyte is furnished by air.

Cathodes, as pulled, weigh 50 to 60 oz. each. After washing, they are melted in a graphite crucible and cast into bars, which have an average fineness of 999.8.

After the gold sand is filtered off, the filtrate discharges into two stoneware tanks 24 by 48 by 24 in. deep. The platinum metals are cemented out with scrap hoop iron. The cemented product is filtered through the type of filter previously mentioned. The residue is thoroughly washed, dried and ground in a small ball mill and shipped to the precious-metals refinery of The International Nickel Co. at Acton, England.

The furnace room contains the roaster, doré and whisker furnaces, which are connected to the same flue system. The flue is outdoors, in the yard. It is 4 ft. in diameter and fabricated out of ¼-in. boiler plate. The brick doré flue enters the system at the extreme end and a straight length of flue 50 ft. long leads into a sheet-iron settling chamber provided with baffles. This chamber measures 12 by 26 ft. by 12 ft. high. The outlet flue runs parallel with the inlet flue and is 96 ft. long. It connects with a 54-in. variable-speed fan, capacity 15,000 cu. ft. of air at 600° F. at 4-in. static pressure. The fan is directly connected to a 25-hp. variable-speed motor.

The fan discharges into a scrubber. Solids washed from the gases collect in the bottom of the scrubber and are discharged through an opening in the bottom into two sumps. The scrubber solution is in closed circuit with these sumps and is pumped up into the scrubber header by a 3-in. variable-speed direct-connected vertical-lead pump.

The solution is circulated until it reaches a density of 15° Bé. When it reaches this point it is pumped through a wooden plate and frame press by means of a 1½-in. duriron pump. The filtrate is then sent to the selenium plant for the recovery of its selenium content. The residue caught in the filter press is discharged into a lead-lined dump car and is transferred to the whisker furnace for further treatment. The scrubber is connected to a 50-ft. lead-lined iron stack 24 in. in diameter by a 30 by 42 in. lead flue, supported by 1½ by ¼-in. steel straps.

The whisker furnace treats all the materials caught in the scrubber. Besides the precious metals, the scrubber sludge contains a high percentage of selenium. The roasting of this sludge converts the selenium to the oxide which collects in the flue chamber in the form of "whiskers."

The whisker furnace is similar in construction to the acid roaster except that it has a firebrick bottom, and the fireboxes and oil burners are at the end of the furnace. A charge consists of one ton of sludge and is rabbled continuously. After all the selenium has been driven off, the calcined sludge is raked out into steel boxes and charged into the doré furnace. The gases from the whisker furnace are led off through a brick uptake, which is connected by a 24-in. diameter steel flue to the "whisker chamber." This chamber measures 14 by 6 ft. by 4½ ft. high and is constructed of ¼-in. steel plate. Six doors cover one side, through which the selenium whiskers are removed. These are raked out at intervals and transferred to the selenium plant for the recovery of the selenium content.

The selenium plant is in the southeast corner of the silver refinery and adjoins the furnace room. It contains a lead-lined storage tank 12 ft. in diameter and 7 ft. deep for scrubber solution, and three lead-lined wooden tanks 6 ft. in diameter and 6 ft. deep, one for dissolving the whiskers and the other two for neutralizing the solution. Each tank is equipped with air agitation.

Scrubber solution or leached whisker solution is charged into the agitators and is neutralized with the alkaline solution from leaching the niter slag. The tellurium is thrown down as tellurium dioxide, and is filtered off through a wooden plate and frame filter press. The neutral liquor is pumped into two lead-lined agitators and the selenium is precipitated with sulfur dioxide gas. After thorough washing with water it is ground, dried, screened and packed for shipment. The process as outlined above is patented and the Ontario Refining Co., Ltd. operates under a license.

Acid-recovery Plant

In this plant the "segregated" solution, or discarded solution from the tank house, is first passed through a nest of 10 tanks carrying a current of 5000 amp., in which are used 25 lead anodes and 24 regular

copper starting sheets. The copper content of the solution is here reduced from one per cent, or 12 grams per liter, to 8 grams per liter. The cathodes from this section are a good commercial deposit and are sent to the wire-bar furnace. The solution is then passed to the second section of 10 tanks carrying the same current, where the copper content

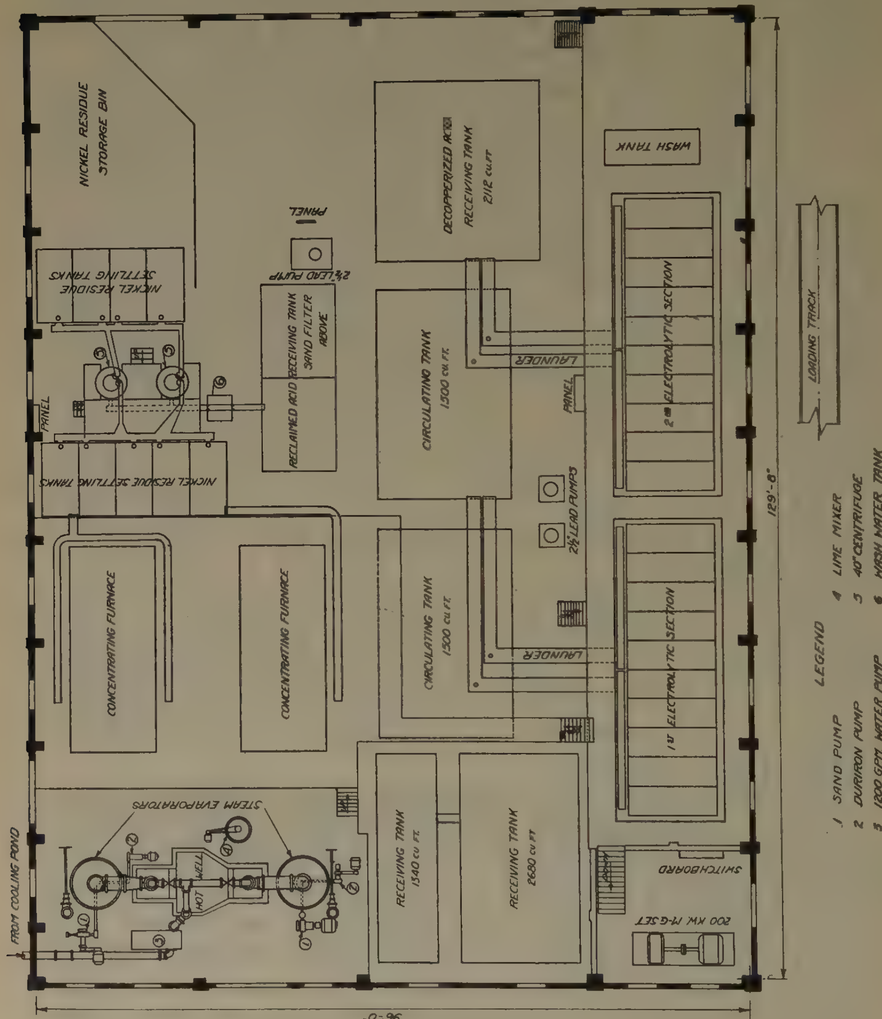

FIG. 8.—PLAN OF ACID PLANT.

is reduced from 8 to 0.2 grams per liter. The cathode copper deposit is dark and loosely adherent and it is delivered either to the anode furnaces or to the silver refinery for cementation purposes.

The "decopperized solution" is next sent to cast lead horizontal tube evaporators (size, 6 ft. 6 in.) where it is concentrated under a vacuum of 28 in. mercury. In these evaporators the solution is heated

by means of low-pressure steam passed through several hundred lead-covered copper tubes. The nickel-acid-bearing solution is concentrated from 23° Bé. to about 60° Bé.; at this concentration considerable impure anhydrous-nickel sulfate crystallization occurs. The evaporator is then dumped and the mixture of nickel residues and acid is delivered by a sand pump to one of the nine iron-precipitating tanks, where the solution is given a period of 24 hr. to cool and settle. By this time precipitation is practically complete, and the clear acid, now of a density of 54° Bé., is decanted off into a storage tank and sand-filtered to be later returned to and used again in the tank-house electrolyte. The nickel residues, which are rather finely divided, are then shoveled into launders and passed into 40-in. suspended centrifuges equipped with rubber-covered steel baskets, where they are washed and dried.

Hydrogen Plant

All lead welding is done with hydrogen and air. The hydrogen is generated in a plant adjacent to the tank house, equipped with a 24-kw. motor-generator set, a bank of eight small-type electrolytic cells, and two 12-ft. diameter gas holders. The hydrogen is produced by the electrolysis of water, using caustic soda as a conducting medium, with iron anodes and cathodes and asbestos diaphragms. Hydrogen of 99.9 per cent purity is generated at the rate of about 100 cu. ft. per hour.

Water System

Services for which water is required in the refinery may be divided into four general classes, each necessitating water with different characteristics from the others, as follows:

1. Boiler-feed water requires soft water free from salts, acid and organic matter.
2. Bosh cooling water requires large volumes of cool but not necessarily pure water.
3. Fire protection and general purposes require a dependable source.
4. Drinking water must be pure and potable.

Therefore the provision of a permanent water supply for the refinery constituted one of the most difficult problems in the construction of the plant. All outside water mains are buried 6 ft. underground.

Water for the first three services is obtained from Kelly Lake, about one mile distant, while drinking water is obtained from the Town of Copper Cliff. Boiler-feed water is softened by the Permutit system in the water-treatment plant. Bosh cooling water is pumped from a cooling pond approximately 350 ft. east of the anode-casting building. This pond, or reservoir, is 350 ft. long, 314 ft. wide, with an average depth of

6 ft.; an area sufficiently large to eliminate the necessity of cooling sprays. It is at a lower elevation than the boshes, from which the water returns to it by gravity. The bosh pumphouse contains three single-stage, motor-driven centrifugal pumps with a capacity of 5000 gal. per min. against a 93-ft. head, and a fire pump with capacity of 1000 gal. per min. driven by a steam turbine for emergency use in case of failure of electric power. The service water supply is obtained from Kelly Lake through a 12-in. pipe line. Two motor-driven two-stage centrifugal pumps are used, each with a capacity of 1000 gal. per min. against a 350-ft. head.

All motor-driven pumps in the bosh pumphouse and at Kelly Lake are operated by remote control from the low-pressure air-compressor room in the casting building. A 2200-volt pole line supplies the pumps, which are started through compensating starters at the bosh pumphouse.

Another reservoir, 175 ft. long by 175 ft. wide, with average depth of 4 ft., is situated at the acid plant for cooling the water supplied to the condensers for maintaining a vacuum in the evaporators.

Coal-pulverizing Plant

The coal-pulverizing plant is in the southwest corner of the property at some distance from the main group of buildings. The incoming coal, which is West Virginia, low-sulfur, gas slake running about 37 per cent volatile matter, 54 per cent fixed carbon, and 6 to 8 per cent ash, with sulfur under 1 per cent, is received by rail in a thaw shed along the east side of the building. This shed is 180 ft. long by 17 ft. wide, which is long enough to accommodate five cars. It is heated in winter by steam coils. The coal is dumped into a track hopper, crushed to ¾-in. size and elevated to a 50-ton raw-coal storage bin 65 ft. above ground level. Any tramp iron left in the coal is removed by electric magnets. Leaving the raw-coal bin, it passes through a rotary drier, fired with pulverized coal, with a capacity of 20 tons per hour. The dry coal is stored in two cone-shaped bins and fed to two pulverizers, each of 6 tons per hour capacity. It then passes through cyclones to two weighing hoppers, from which it is pumped as required, by compressed air, to storage hoppers at the furnaces and the auxiliary boiler. The air-electric valves to the bins in the delivery system are controlled from a panel in the pulverizing plant, and the filling of the bins is practically automatic. The coal is transported a maximum distance of 1300 ft. in an overhead line insulated together with a heating steam line. A motor-driven 350 cu. ft. air compressor provides the air for delivery of the coal. Each bin has a steam connection for smothering in case of fire. All equipment and steel work in the coal plant is painted with aluminum, on which any coal dust can be readily seen. The plant is vacuum-cleaned each day and kept in a thoroughly clean condition.

Steam and Power

Large quantities of steam at various pressures are required for the different processes. There are large demands for low-pressure steam for heating tank-house electrolyte, for evaporators, drying ovens, water heaters, and for heating buildings. This is particularly true because of the cold winters and the necessity for uniformly warm temperatures in the electrolytic process buildings. High-pressure steam is required for the turbine-driven water pumps, reciprocating vacuum pumps, and a large number of steam siphons, etc., and the excess is used to drive a turbine generator for supplying power demands over and above the purchased peak load.

The bulk of the steam is generated by waste-heat boilers, of which there are six, one over each of the three anode furnaces and the three wire-bar furnaces. An auxiliary boiler is at the north end of the cathode-storage building, centrally located between the anode furnace and wire-bar furnace boilers. Space has also been provided here for a second unit if it should be required in the future. The auxiliary boiler is brought into use when waste-heat steam is insufficient to supply the demand.

The waste-heat boilers are of the Stirling, water-tube, four-drum type, and each has a heating surface of 6500 sq. ft. The design is standard, except for one row of slagging tubes installed in the front row of the first pass. The boilers are provided with economizers and superheaters, the latter giving 110° F. superheat. The safety valves are set to relieve at 180 lb. gage pressure. The steaming rate ranges from 40,000 lb. of steam per hour during the melting down to 7000 lb. per hour during the refining period. With two or more waste-heat boilers in operation, sufficient steam is provided for plant requirements except during casting periods.

The auxiliary boiler is also of Stirling type and has a heating surface of 10,000 sq. ft. It supplies steam of the same quality as the waste-heat boilers, but is fired by two horizontal pulverized coal burners. Flexible fuel-oil nozzles are also provided for emergency operation and fire protection in case of failure of other sources of power and steam.

The main source of low-pressure steam is the exhaust from a 2500-kva., at unity power factor, single reduction geared turbogenerator, generating power at 2300 volts in the power house. This turbogenerator supplies all the load above the amount of power contracted for and exhausts against a back-pressure of 10-lb. gage. Automatic regulating valves and a de-superheater are provided so that high-pressure steam can be bypassed around the turbine into the low-pressure line when required.

The power house is at the northwest corner of the tank house, so placed to be close to the electrolytic circuits to reduce direct current

transmission losses. In the event of extension of the tank house, it would occupy a central position. Normally, the two sources of power (manufactured and purchased) are connected in parallel, the lighting and power load being carried by the turbine, but also, in part, on purchased power whenever there is an excess over the electrolytic load requirements. At the alternating-current switchboard, a load regulator holds the purchased power at any predetermined constant, and so controls the turbine as to take care of all peaks above this amount. Power is purchased at 30,000 volts and is stepped down to 2300 volts through a bank of three 2000-kva. transformers, with a fourth as a spare. All are connected to the primary and secondary buses by double-throw disconnects.

From the secondary busbars, a pole line leads to the acid plant to provide power for a 2000-kw. electric steam generator operating at 2300-volts, while underground cables lead to the power house and connect to the busbars at the purchased-power switchboard. At this switchboard the 2300-volt power supplies three synchronous motors, which drive the three motor-generator sets supplying the electrolytic load. Each set consists of one 2880-hp., 500-r.p.m. synchronous motor in the center, driving two 6000-amp., 80 to 160-volt d.c. generators, which are operated in parallel with direct-connected exciters. Each set is rated at 2500 kva. The third generator is a spare and may be used interchangeably with the other two in a very flexible manner. Clean air for the power house is drawn in by two airplane propeller-type fans through oil-treated movable filters.

Compressed air is supplied for use throughout the plant by a 2000-cu. ft., vertical, two-stage air compressor driven by a 400-hp. synchronous motor at 2200 volts. A 500-cu. ft. horizontal, two-stage compressor is provided as an auxiliary.

The power house is equipped with a 15-ton hand-operated overhead crane.

Power, steam, air and fuel are metered to each service and each department is charged for what it uses.

Laboratory and Mechanical Department

The laboratory occupies the top floor of the administration building and is 85 ft. long by 41 ft. wide, divided into modernly equipped departments for the control of all plant operations. Electric furnaces, muffles and hot plates are used throughout. All hoods have individual fans, which provide a clear gas-free atmosphere at all times.

The refinery, being built a long distance away from manufacturing centers, found it necessary to provide a shops building and equip it with the most modern machinery in order to take care of any emergency. This building is 400 ft. long and 70 ft. wide, and is subdivided into carpenter, pipe, electrical, machine, blacksmith, ironwork and welding

shops. The transportation and handling systems are so arranged that each shop operates independently of all others.

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The Nichols Series System of Electrolytic Copper Refining

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The Nichols series system of electrolytic copper refining has successfully maintained its strong competitive position in the copper-refining industry of today; therefore the improvements made in accomplishing this result are of interest. While additional interest would lie in specific comparisons of results obtained in series operation with those in multiple, we have refrained from such a comparison because it can be fair to both only when they are operating under similar conditions on the same grade of materials.

Historical

The Nichols Copper Co. started a small refinery at Laurel Hill, using its present system, about 1892. The details of the process in its early stage of development were kept strictly secret, so that no information was published or interchange of ideas permitted with the staffs of other copper refineries, where the multiple system was being developed at the same time. However, the fundamentals of the process and type of cell used today differ very little from their earliest form. All operations including loading and unloading of tanks, and all transportation as well, were carried on by hand. The plant remained in this condition with a gradual expansion taking place until the World War forced such a rapid development that little time or thought could be devoted to modernization or mechanical improvement requiring careful design or detailed planning. Considerable progress was made, however. (1) Large refining furnaces were developed. (2) The old expensive air-lift type of circulation was abandoned in favor of the present system. (3) Operating technique was gradually developed to a high point of efficiency. (4) Tank construction and lining were greatly improved, the first concrete tank being installed in 1915. (This tank is still in operating condition.) (5) A new power plant was constructed. (6) Fuel oil was substituted for lump coal on the refining furnaces. (7) Mechanical casting of the finished product was accomplished.

During this entire period the plant was particularly fortunate in receiving only high-grade, clean copper bullion coming principally from the Phelps Dodge mines in Arizona. This, together with the lack of

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publicity given to the process and the fact that the only other series system in operation used a rolled anode, contributed to the erroneous impression that the series system could successfully treat only high-grade, clean copper bullion.

The keen competition and high wages that developed at the close of the World War focused attention on operating costs and led to an extensive mechanicalization period.[1] Near the beginning of this period the treatment of lower grade, high-impurity blisters and secondary materials was undertaken. At first, owing to lack of experience with this type of material, many difficulties were encountered. (1) The old type of anode furnace with a sand bottom and silica brick sides proved so unsatisfactory that it was finally abandoned, and the present type, using silica-brick bottom, magnesite-brick sides and silica-brick arch was adopted. (2) It was also found advantageous to carefully blend the different grades of copper to make a charge as uniform as possible and then refine the metal much more thoroughly than had ever been required with only high-grade bullion. This operation is now carried on in a large, chute-charged, melting furnace, which feeds two casting furnaces, each of 100 tons capacity. (3) As the cell spacing in the electrolytic tanks proved to be too close to permit proper slimes settling, it became necessary to reconstruct them with new caps to provide wider spacing. Cars and crane racks also had to be revised to conform. (4) It was found necessary to regulate carefully electrolyte impurities within certain limits. (5) The use of addition agents was improved. (6) An improved type of electrolyte heater and larger feed lines were installed, making it possible to operate with more circulation on the tanks and more accurate temperature control. (7) Tank construction was further improved to provide more efficient insulation and reduce power losses. (8) As it was then found practical to treat a heavier anode, its weight was gradually increased until today it is 25 per cent heavier than the one used up to 1924. (9) Wherever the nature of the work or the installation of mechanical equipment made it possible, piece work has been established. All of these factors have contributed materially to improve efficiencies, reduce anode scrap to a minimum, eliminate waste and reduce the amount of labor required.

In 1926 a completely new wire-bar furnace unit was installed. This equipment consisted of a charging crane, a 40-ft. Walker-type casting wheel (which more recently has been revised to pour five bars at a time using a single ladle), a new waste-heat boiler and a bosh-water cooling and recovery system

A new copper sulfate plant for treating the waste solution from the tank house was put into operation in 1930. This is a thoroughly modern

[1] M. H. Merriss: *Trans.* A.I.M.E. (1926) **73**, 123.

F. R. Corwin and C. S. Harloff: *Trans.* Amer. Electrochem. Soc. (1930) **57**, 231.

plant with up-to-date handling and pumping facilities, modern screening equipment, lead-lined, steel crystallizing tanks and oxidizers, and mastic floors.

Throughout the plant, handling facilities have been greatly improved, with a corresponding reduction in labor, by the installation of cranes where practicable, and monorail cranes, which are peculiarly well adapted for the service desired in this particular plant. Tractors, Fordsons and battery locomotives have materially reduced interdepartment transportation expense.

A careful study of steam and power requirements has resulted in many economies in both generation and distribution, which have materially cheapened these major items of supplies. The chief benefit has been derived from systematic and thorough cleaning of both external and internal heating surfaces of boilers while in operation as well as during shutdown periods, resulting in boiler efficiencies continuously in excess of 80 per cent. The same treatment applied to waste-heat boilers, together with the fact that they are over the furnaces and receive radiant heat direct from the molten metal bath, combine to give the boilers a relatively high efficiency, although not equal to that of direct-fired units.

Tank House

The standard electrolytic cell consists of a wooden or concrete shell 4 in. thick, mounted on lead-capped, tile piers set in a concrete floor, which is protected with a layer of mastic 2 to 3 in. thick. The cell has a 2-in. bottom lining made from the following materials: blown oil, 500 lb.; T.R.A. asphalt, 300 lb.; powdered silica, 600 lb.; sea sand (40 mesh), 1800 lb. The side lining is ¾ in. of blown oil mopped on in successive layers, which are burned on with a hot iron. The completed tank is 16 ft. by 5 ft. 4 in. by 5 ft. 2 in. deep inside the lining. It is capped with stationary wooden spacing blocks on each side forming slots into which the 102 iron suspension bars carrying five anodes each can fit. Each tank, therefore, contains, when loaded, 510 regular anodes, weighing about 56,000 lb., and 22,000 lb. of electrolyte. In addition, a positive copper busbar 2½ by ¾ in. spanning the width of the tank is connected by copper cable to the negative bar on the tank immediately preceding it. From it five anodes at 150 lb. each are suspended by copper links. These serve to introduce the current into the cell. At the negative end, a similar busbar carrying five depositing cathodes at 45 lb. each conducts the current out of the cell. The anode spacing is 1$^{3}/_{16}$ in. center to center with 1-in. clearance on the side of the tank and ½ in. between adjacent anodes on the same suspension bar. There are no busbar connections throughout the tank. Each anode serves as a bipolar electrode receiving a deposit of pure copper on one side while the impure anode copper is being dissolved from the opposite side. The silver and

gold slimes thus liberated gradually fall off and accumulate in the 5-in. space under the electrodes. The tank is further equipped with a copper cut-out bar for bypassing the current during periods of loading, unloading and repairs. A 2-in., hard rubber plug for draining electrolyte is provided in the bottom of the tank, which feeds into the electrolyte launders in the floor between the tanks. Also, a hard rubber overflow equipped with a reversible lead cup for controlling the height of solution in the tank is in the center of one side near the top. The tanks are electrically connected in groups of about 44 (all groups are not uniform) in two parallel

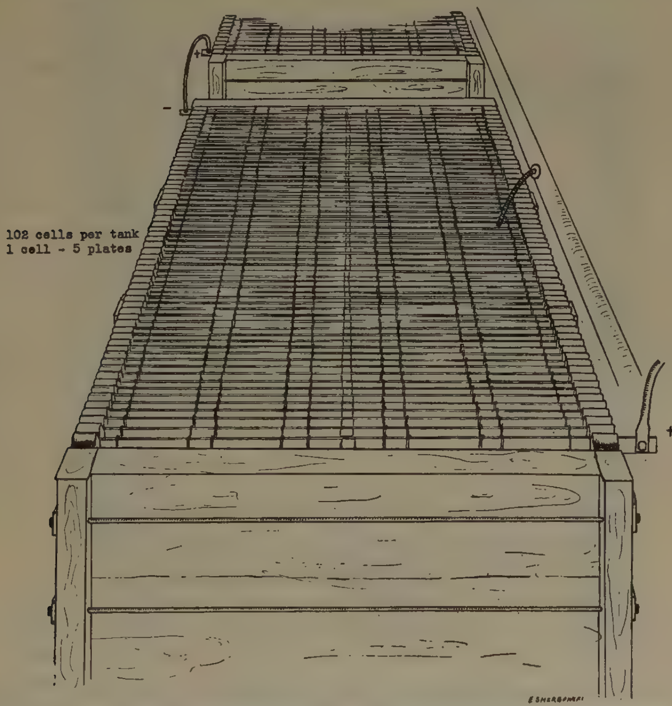


FIG. 1.—PLAN VIEW OF LOADED SERIES TANK, SHOWING SET-UP, SPACING BLOCKS AND CIRCULATION INLET, NICHOLS COPPER CO.

lines of 22 each. This arrangement serves two purposes. It gives a power load of desired characteristics to be carried on the rotary converters that supply current to the tank house. It also furnishes a sensitive indicator of typical tank house troubles affecting the total resistance of either or both legs, such as short-circuiting, ground losses, burned out busbar plates, loose or dirty connections, deficient electrolyte and insufficient circulation, which result in uneven current distribution and excessive power consumption. The tank house contains 417 of these tanks, which can produce up to 35,000,000 lb. of cathodes per month, depending upon the current density used. They are loaded and unloaded by 5-ton cranes equipped with a lifting rack that will pick up 17 bars or 85 anodes

to a draft, thus requiring 6 drafts per tank. All tanks are connected so that the direction of current flow is the same for the entire tank house. This arrangement eliminates the possible error of loading the anodes to receive the cathode deposit on the wrong side. Fig. 1 shows a typical tank and its arrangement in series with other tanks; Fig. 2, the cross-section through a loaded tank.

Electrolyte for circulation is stored in lead-lined, wooden receivers at sufficient elevation to provide gravity flow to the tanks. From the

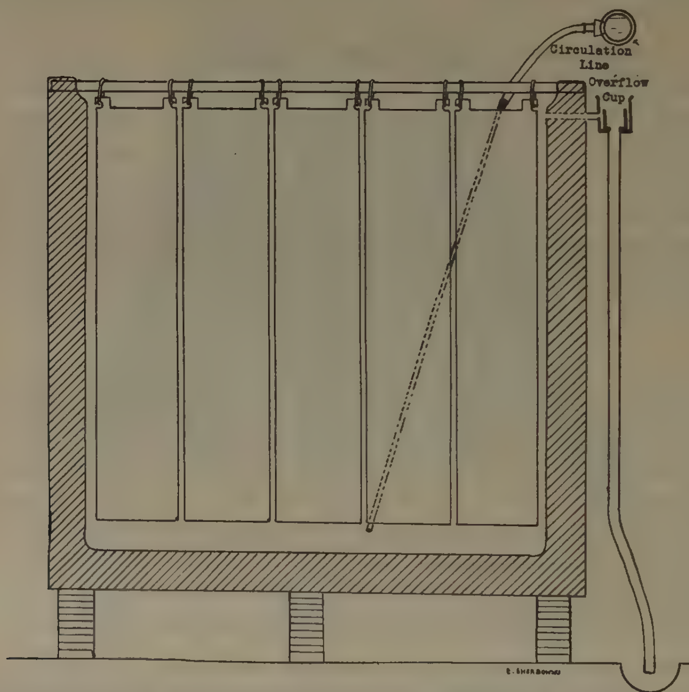


FIG. 2.—ELEVATION OF ELECTROLYTIC TANK SHOWING PLATES SPACED, CIRCULATION GLASS AND OVERFLOW, NICHOLS COPPER CO.

receivers, the solution flows through lead-lined, heat exchangers into a 12-in. manifold from which it flows into the lead 4-in. main feed lines. At proper intervals, flanged lead nipples are burned to these feed lines. To the nipples are bolted hard rubber connecting tubes to which rubber hose connections can be attached by hand. A glass tube carries the solution at the rate of about 3.5 gal. per minute to within 5 to 8 in. of the bottom of the tank. The electrolytic action promotes additional circulation within the tank itself. Solution overflowing from the tanks is carried by launders in the floor to wooden, lead-lined, settling pits from which it is pumped back into the receivers. The temperature of solution in the tanks is maintained at about 122° F. For control purposes the tank

house is divided into four circulation systems which are cross-connected to permit interchange of solution.

The hand-cast anode, as previously mentioned, weighs about 110 lb. It is made from a wide variety of blister coppers as represented in Table 1, showing the composition of material treated during the past 10 years. It is 56 by 12 by 9⁄16 in., with two ears forming the top part, which subsequently are punched out to form the suspension lug. It is suspended from the steel hanger bars by acid-resistant, cast copper-alloy links, and the anode composition varies between the limits shown in Table 2.

TABLE 1.—*Typical Analyses of Blister Treated at Laurel Hill*

Brand	Cu, Per Cent	Ag, Oz. per Ton[a]	Au, Oz. per Ton[a]	As, Per Cent	Sb, Per Cent	Pb, Per Cent	Ni & Co, Per Cent	Zn, Per Cent	Fe, Per Cent	S, Per Cent	Se & Te, Per Cent
Katanga	96.50	51.80	0.015	0.201	0.009	0.40	0.038	0.20	0.30	1.09	0.0002
Rio Tinto	98.79	18.40	0.33	0.158	0.019	0.069	0.040	0.017	0.08	0.109	0.0065
Pyrite	98.60	46.00	0.04	0.142	0.015	0.240	0.044	0.01	0.044	0.181	0.002
Magma	99.15	59.20	0.60	0.045	0.025	0.035	0.034	0.002	0.022	0.032	0.0135
U.V.X.	99.15	17.90	0.36	0.025	0.008	0.007	0.035	0.002	0.045	0.05	0.15
Granby	99.25	32.50	1.10	0.003	0.005	0.025	0.065	0.002	0.030	0.15	0.025
Phelps Dodge	99.20	25.00	0.50	0.005	0.006	0.03	0.04	0.002	0.04	0.035	0.03
Tennessee	99.20	8.50	0.05	0.005	0.006	0.013	0.025	0.002	0.044	0.023	0.025
Ducktown	99.15	5.60	0.05	0.0044	0.0045	0.047	0.093	0.002	0.053	0.074	0.016
Bor	99.10	3.50	0.96	0.0022	0.004	0.003	0.012	0.002	0.080	0.012	0.0007
Noranda	99.20	14.00	3.10	0.004	0.004	0.005	0.025	0.002	0.025	0.151	0.20
Nichols blister	96.00	15.00	0.70	0.20	0.15	0.30	0.80	0.01	0.05	0.10	0.02
Secondary material	93.00				0.5	3.00	1.0	3.0	2.0		

[a] One ounce per ton is equivalent to 34.3 grams per 1000 kilograms.

TABLE 2.—*Anode Composition*[a]

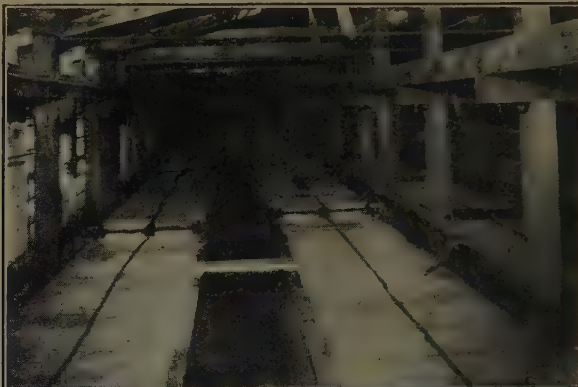
Copper, per cent	99.20–99.60	Selenium and tellurium, per cent	0.02 –0.17
Oxygen, per cent	0.10– 0.30	Nickel and cobalt, per cent	0.06 –0.27
Arsenic, per cent	0.03– 0.144	Iron, per cent	0.001–0.0026
Antimony, per cent	0.01– 0.06	Silver, oz. per ton	10 – 48
Lead, per cent	0.02– 0.12	Gold, oz. per ton	0.2 –2.4

[a] With high amounts of the insoluble impurities present, there is some loss of efficiency due to uneven anode corrosion, but this is no more pronounced in series electrolysis than in multiple.

Following the casting and partial cooling, the warm anodes are punched, carefully straightened, assembled on the suspension bars and spray-painted (Fig. 4) on the side that is to receive the pure copper deposit, with a hot (180° F.) 25 per cent sodium resinate solution, which forms a cleavage zone that facilitates the removal of anode scrap without materially increasing the electrical resistance at the depositing surface.

After a tank is loaded, it is carefully inspected and the anodes adjusted to see that all are hanging true without touching each other or the side

of the tank at any point. Electrolyte is then fed in by a 2½-in. hose attached to a valve connection in the main circulating line until 6 in.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4


5

Fig. 3.—One series of concrete tanks.
Fig. 4.—Painting anodes with sodium resinate solution.
Fig. 5.—Transferring anodes from punching and painting room to tanks.

of the anodes is submerged. The cut-out bar is then disconnected and the current begins to pass through the tank. Filling with electrolyte continues until solution runs out the overflow; after that the filling hose is

shut off and only the regular circulation is maintained. This procedure for cutting in tanks is carefully followed in order to obtain an initial cathode deposit on the resinous paint as quickly as possible. If it remains exposed to the hot electrolyte too long it will be dissolved and the cathode deposit will knit so tightly to the anode that the remaining anode scrap left undissolved at the end of the cycle cannot be removed.

Pulling schedules leave the tanks in circuit long enough to theoretically convert all anodes completely to cathodes. With anodes of the present thickness, about 10,500 ampere days per tank are required. The number of days in circuit is determined by the current density employed. This tank cycle has been varied from 13 to 34 days in order to meet the production rates desired. While it is impossible to obtain a complete conversion of anodes to cathodes, the percentage of anode scrap can be kept extremely low. During the past four years it has never exceeded 8 per cent. The anodes are sufficiently uniform to attain this result without excessive resolution, as evidenced by the figures of power consumption based on power-plant switchboard readings (Table 3). These results represent the entire production of the tank house for at least one month. The large increase in cathode yield below 19 amp. per square foot is due partly to an increase in weight of the anode used when these results were obtained.

TABLE 3.—*Power Consumption*

Current Density, Amp. per Sq. Ft.	Cathodes Used, Lb. per Kw-day	Current Density, Amp. per Sq. Ft.	Cathodes Used, Lb. per Kw-day
13	471	22	297
13.8	450	23	289
16	410	24	276
17	400	25	272
19	328	26	270
21	308		

Inspection and short-circuit work play a prominent role in obtaining good efficiencies. Approximately two men are used on this work for every 3500 tons of monthly cathode production. (1) Careful inspections are frequently made of all tanks in operation. (2) Voltages are checked daily on each tank. (3) Group ampere readings are taken hourly to insure proper current distribution on the two legs. (4) Circulation measurements are made daily and tank temperatures are checked at the same time. (5) The men have been trained to detect the growth of copper on the bottom of the tanks, which is peculiar to series system, and to break it up so that it will be at least partially redissolved without interruption to the tank's operation. (6) Anodes that have become short-circuited can also be lifted and the cause removed without any interruption to the tank's operation.

Electrolyte control is another important factor in obtaining good efficiencies. Electrolyte temperature is regulated to maintain the tanks at about 122° F., as the most economical point. While it may be possible to obtain lower power consumption at a higher solution temperature, still the increase in steam consumption and the fact that hot electrolyte softens the tank lining and causes it to slip more rapidly militates against the use of higher temperatures. Based on a yearly average, approximately 500 lb. of steam at 12 to 15 lb. pressure is used per ton of cathodes.

The composition of the electrolyte is regulated by the addition of concentrated sulfuric acid fed slowly into the settling pits and of water partly supplied by cathode washing. These additions are balanced by evaporation and the daily withdrawal of solution highest in impurity to the copper sulfate plant. The loss of copper is made up by the chemical solution of half of the copper content of the suboxide present in the anodes. By careful regulation it is possible to avoid the use of depositing-out cells or lead grids. The copper content of the electrolyte is maintained at 2.75 to 2.85 per cent and the free sulfuric acid at 17.50 to 18.50 per cent.

Cathodes of the desired purity have been produced with electrolyte up to 1.0 per cent nickel, 0.35 per cent arsenic, 0.06 per cent antimony, and 0.05 per cent iron. Chlorine as sodium chloride is added daily as required to keep the electrolyte at about 0.0003 to 0.0005 per cent chlorine. Glue and Goulac are added at a ratio of one ounce and one pound respectively for every 50 tons of cathodes deposited. These figures for addition agents are average, of course, and must be varied somewhat as the character of deposit indicates the necessity.

At the completion of a run, the tank is emptied of solution and the cathodes are washed in place, to remove all slimes, with a long-handled spray using about 10,000 lb. per tank of condensate water at 100° F. and 120 lb. pressure. This water also flows into the electrolyte circulation system, partly replacing the evaporation from the operating tanks. The cathodes are then removed from the tank, loaded on rack cars, and transported to the stripping room.

Before the tanks are reloaded with anodes for the next cycle, the lining of each tank is inspected and any soft spots or cuts are repaired. As there is always a little slippage of the side lining, it is necessary periodically to shave it off toward the bottom and renew it near the top of the tank, to maintain its correct size. With proper maintenance, the life of the wooden tank is about seven years, and the life of the concrete tank appears to be indefinite.

In the stripping room each carload of cathodes is placed on the floor by crane. The men carefully strip off the remaining anode scrap, which is returned to the casting department. They also load the clean cathodes on flat cars ready for charging into the refining furnaces, or for sorting

and cutting for shipment, as the case may be. The labor required for this operation is approximately 0.7 man-hour per ton of cathodes.

Following the removal of the washed cathodes from the tanks, the silver slime is pailed out into wooden, lead-lined boxes mounted on cars, which transport it to the treatment plant. A coarse screen (quarter mesh) removes all copper nodules and foreign material, after which the slime is pumped as a slurry to storage tanks that feed an Oliver filter. The filtrate is returned to the electrolyte system as make-up water while the filter cake is fed to a Nichols-Herreshoff roaster, which dries the slime and oxidizes the metallic copper present. The roasted slime is leached to recover the copper and the leach liquor goes to the copper sulfate plant for further treatment. The leached slime is washed, dried and barreled for shipment to another plant where the gold and silver are finally recovered.

Electrolytic Copper Refining at Mount Lyell, Tasmania

By R. M. Murray,* Queenstown, Tasmania

Although blister copper has been produced at the Mount Lyell Mining and Railway Company's works since 1896, it is only during the past five years that electrolytic refining has been undertaken locally, this having been done previously at customs refineries, at first in the United States, and afterwards at Port Kembla, New South Wales.

Fig. 1.—General view of portion of works.
Left to right: smelter building, converter building, blower building (in which converter-generators are housed), tank house and anode dock. Wire-bar furnace building in background at center.

Possessing a hydroelectric power plant capable of generating power considerably in excess of the requirements of its mines and works at the time, the company was in a favorable position to undertake its own refining work, provided this advantage was not offset by the high cost of coal, which would have to be imported from the mainland, and provided also that the capital cost of the plant would not be unduly large in regard to the relatively small amount of copper to be treated. A study of these several conditions led to the adoption of a high-density plant using blister-copper anodes and equipped with an electric boiler to provide any

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heating required. The original installation was designed by A. E. Wheeler, of New York, with a capacity of 8000 long tons per annum, and this has since been enlarged to a capacity of 13,000 tons per annum.

The plant itself and the method of operating present some unusual features, and in submitting the following brief description acknowledgment is made of the assistance received from Mr. W. Hart, Refinery Superintendent, and his assistant, Mr. D. H. Wilsdon.

Tank House and Tanks

The tank house contains 144 cells arranged in 24 cascades of five cells each and four cascades of six cells, the former being for the deposition of

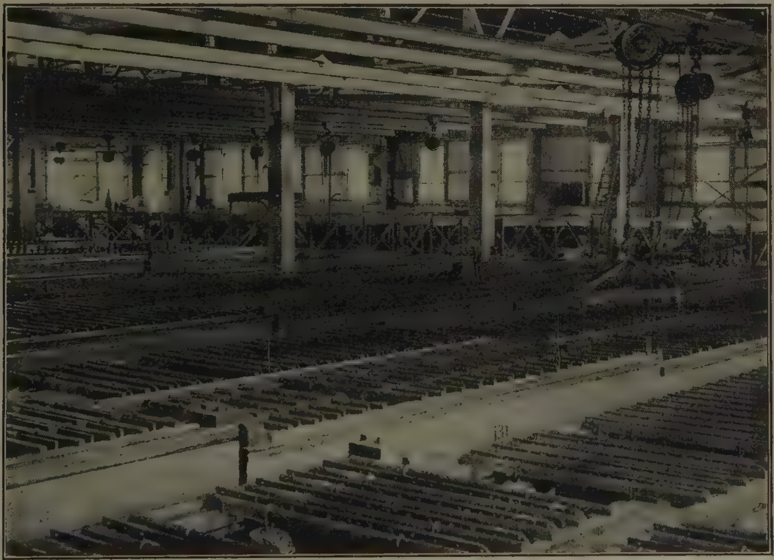

Fig. 2.—Interior of portion of tank house. Concrete stripper cells in foreground.

commercial cathodes and the latter for making starting sheets. The building, which is 132 by 145 ft., is of steel-frame construction, painted with acid-resisting gray bituminous paint, and roofed and enclosed with corrugated fibro-cement, and is liberally provided with movable windows, placed in the sides, and monitors. A concrete wall separates the cell room from the purification and slime-treatment division, and a spacious basement extends the full length of the building, giving ample head room under the cells, and providing room for the various electrolyte receiving and heating tanks and circulating pumps. The floor is of concrete, covered with one inch of bitumen mastic except between the cells, where the wooden walkways are supported from the cell piers. A space 40 ft. wide over the whole length of the building in front of the tanks is used for

racking anodes, stripping and preparing starting sheets and for handling and washing cathodes. This is spanned by a 5-ton electric traveling crane, but there is no crane over the tanks, which are operated by ½-ton chain blocks suspended from I-beams attached to the roof principals, as described later.

The original tanks are constructed of Douglas fir lined with 6 per cent antimonial lead weighing 5.7 lb. per square foot. Those added later are of the same dimensions; viz., 10 ft. 3 in. by 2 ft. 9 in. by 3 ft. 9 in. deep, but are built of reinforced concrete, with the interiors thickly coated with bitumen before lining with the same class of lead. Weepholes are provided in the concrete bottoms for the purpose of localizing leaks. This type of tank has proved to be highly satisfactory, the maintenance costs are less, and the lead lining shows no tendency to creep or bulge away from the walls as do wooden ones.

The copper conductor bars are 6 by 2 in. in section, and will carry a maximum current of 11,000 amp. All permanent joints on the tank busbars are soldered, and the connecting conductors between cascades, which are 7 by 2 in. in section, are U-shaped, with the horizontal part beneath the walkway.

Anodes

The anodes are cast in a straight-line machine from blister copper poured direct from the tilting ladle filled from the converter vessel, those for the commercial cells weighing 600 lb. and those for the stripper cells 700 lb. Copper-wire suspension loops ¼ in. in diameter are cast in the top corners, which project beyond the main mass of the anode. These take the place of lugs; and to prevent them from burning through at the metal line, they are carefully covered with lime or cement. The ends are turned up to prevent their pulling out while the anode is in suspension in the tanks. The standard anodes are 3 ft. 0¾ in. long by 2 ft. 3½ in. wide, and vary from 3 to 2½ in. in thickness from top to bottom. Those for the stripper cells are 1½ in. deeper and 3 in. wider.

Blister copper carrying 0.1 per cent sulfur has been found to give the best type of anode. This slightly underblown copper is denser and gives a more even surface on cooling than the more completely blown, and the tendency for nodular formation on the cathodes is reduced. Copper blown less is too porous, and corrodes unevenly, producing rough cathodes. Anodes made from overblown copper have been experimented with, the converter vessel being practically used as a fire-refining furnace with the formation of cuprous oxide, which acts as a scorifying agent. No attempt however was made at poling, as this was not practicable under existing conditions. The improvement in the anode slimes from this class of anode was most marked, the concentration of silver and gold being doubled because of the impurity elimination in the last stage of

converting. The disadvantages were serious, however, for although the bulk of the copper was very dense, big cavities formed in the castings and the surfaces were composed of very large blisters, up to 6 in. in diameter. Very uneven corrosion resulted and large flakes of copper occasionally became detached and formed short circuits between anode and cathode, with serious loss in ampere efficiency.

Variation in the weight of the anodes is large, with the result that there is a considerable difference in the times occupied in their corrosion. This precludes the withdrawal of all the anode scrap at one time, and each anode has therefore to be handled separately both into and out of the tank, being transported by a ½-ton chain block suspended through bell insulators from trolleys that run on I-beams placed lengthways over the cascades, the usual method of loading and unloading the tanks by means of a crane spanning the cells not being practicable when using anodes of this class. This point is made clear by the fact that during the period between successive loadings, which varies from 18 to 21 days according to the current density, it is usually found necessary to replace 40 per cent of the anodes with oversize scrap from tanks previously cleaned.

The average amount of the final scrap amounts to 12 per cent of the original anode, and this is remelted in the converter vessels, being added after blowing is well advanced and the slag has been poured off. At this stage the vessel is hottest and the cooling effect of the scrap is beneficial.

Starting Sheets

Tough and smooth starting sheets of first-class quality are made at a current density of 17 amp. per square foot, this lower density being provided by placing pairs of stripper cells in parallel. Sheets are stripped every 24 hr., and their average weight is 8 lb., the proportion of faulty sheets being negligible. They are trimmed in a guillotine and looped with a Morrow clip machine. Nine per cent of the total quantity is cut for loops, which do not require any annealing.

The acid concentration of the stripper solution is kept between 65 and 75 grams per liter, and its purity is maintained by the regular discarding of solution to the commercial section. Temperature is given close attention, and is maintained at 40° to 44° C. The amount of glue in the circuit is another matter that calls for particular care, 1½ oz. being added each 8-hr. shift. The glue is dissolved in hot water and distributed evenly over the cells by discharging it at a regular rate from a container carried by a man walking between the cells, this method of addition giving more satisfactory results than are obtained by adding a portion at the end of each tank, or the whole in the supply tanks. The glue toughens the sheets and prevents linear crystal growth, which causes brittleness.

The stripping mixture used on the blanks is composed of inexpensive black grease and compounded cylinder oil thinned with kerosene, the consumption of this mixture being one gallon per day for 500 starting sheet blanks. In seasoning new blanks, the mixture is put on thickly with a brush after the blanks have been thoroughly cleaned with fine emery, and no attempt is made to obtain good sheets from the first two drawings.

In stripping, five alternate blanks are drawn from the cells with a five-pair hook bail with 10-in. centers. The blanks are then run to the end of the cascade, placed in a rack and stripped and oiled; then swabbed to remove any surplus oil. Grooved wooden strips are placed on the sides of the blanks to prevent the copper from depositing around the edges, but no bottom strip is used, so that the sheets are stripped in pairs joined at the bottom, these being separated by trimming in the guillotine.

Cathodes

Three-day cathodes are made at a current density varying between 26 and 33 amp. to the square foot, according to the output required. The temperature of the electrolyte is maintained at between 50° and 55° C. and the acid concentration at between 110 and 150 grams per liter, according to the current density. The acid is kept low in order to cause the heat generated by the resistance of the solution to provide practically the whole of the heating of the electrolyte. Glue is added at the rate of one pound per day, at 4-hr. intervals. Sulfonated castor oil is added with the glue to the extent of ¾ gal. per day, the method of its introduction into the circuit being that used with the stripper tanks. The soluble oil evidently has a marked effect in rounding off the nodules. Any excess of glue leads to trouble, in that small nodules protrude from the cathodes and cause treeing, and the cathodes become rough.

The ampere efficiency averages 95 per cent at 27 amp. to the square foot, and 92 per cent at 33 amp. per square foot.

The cathodes are drawn four at a time, by means of a four-pair hook bail suspended from a chain block, and are replaced by starting sheets that have been straightened by being flapped on a steel plate. This operation is carried out without short-circuiting the cells. The cathodes are run to trucks with rack sides, and lifted twelve at a time by a 5-ton overhead crane, by which they are immersed in boiling water for washing, and then racked in wooden stands where loose nodules are removed. They are then weighed and stacked ready for transport.

It has been found advantageous to insert pine slats (1 by ¾ in.) between new anodes and starting sheets until after the two first drawings of cathodes. This insures uniform spacing until the rough surfaces of the anodes disappear. After one day's deposit the sheets are lifted on to a movable inclined flapping board suspended from the chain blocks,

and any irregularities are straightened out. The combined results of the spacing sticks and this extra straightening of the partly grown cathodes has increased the ampere efficiency by 4 per cent.

The elimination of the soluble and insoluble impurities is over 98 per cent. The cathodes are of excellent quality, producing wire bars with a conductivity of not less than 101 per cent.

The cathodes are sent to the customs works of the Electrolytic Refining and Smelting Co. at Port Kembla for wire-bar furnace treatment, these works being situated near the southern coal fields of New South Wales and adjacent to wire and tube works of the Metals Manufactures Co., which absorb a portion of the output. The remainder of the output is shipped to the English market.

A wire-bar furnace was erected at Mount Lyell at the same time as the electrolytic plant, but has not been placed in commission, being kept as a stand-by while the arrangement for customs treatment remains satisfactory.

Anode Slimes

The slimes are of much lower grade than those obtained from refined anodes, being high in copper, lead, sulfur, selenium and tellurium, and low in silver. The quantity of slimes produced is 1.3 per cent of the cathode copper.

The slimes are washed through the plug-holes in the cell, and led by short hoses to U-shaped lead-lined launders that flow to a central tank into which they discharge through a stainless-steel screen punched with holes $\frac{1}{16}$ in. in diameter, which removes the larger pieces of copper. From this tank they are gravitated into a lead-lined blower case, from which they are forced by compressed air to a filter press with hard lead plates, and thoroughly washed until they are quite free from sulfate and acid. They are then dried on a steam table until the moisture content is reduced to 10 per cent, sampled by the usual cone and quarter method, and packed in discarded steel oil containers, which are resealed with the oxyacetylene torch, and dispatched to the customs silver refinery at Port Kembla, New South Wales, for final treatment.

Electrolyte Purification, Etc.

The chief soluble impurity is nickel, which is kept below 10 grams per liter by a daily discard of the electrolyte to the purification section. Here the solution is heated to 70° C. and sprayed intermittently over starting-sheet trimmings in an oxidizer tank. Steam and air are blown through the trimmings to assist the oxidation, and this is continued until the free acid content is reduced to less than 10 grams per liter. The use of pure electrolytic copper, instead of the usual blister copper

shot high in sulfur, is probably unique, but has proved successful in this operation.

The solutions are concentrated in a lead-lined boiling tank fitted with four hard lead steam coils until a specific gravity of 1.47 is reached, then they are run into semicylindrical crystallizing vats and subjected to three days' cooling.

Ordinary starting sheets are used for the deposition of the copper sulfate crystals, these being trimmed to the shape of the tank, and suspended from wooden hangers. About half the deposition takes place on these, and the remainder on the sides and bottom of the tanks. The recovery of bluestone is 85 per cent, and that which deposits on the sheets and tank sides is of commercial quality, some of it being marketed.

The copper remaining in the mother liquor is recovered in a cementation vat filled with scrap iron through which steam is blown to aid the deposition, and only a trace remains in the final discard.

One liberator cell in circuit all the time and a second for one-third of the time are sufficient to deplete the requisite amount of copper from the electrolyte, and disintegrate the bluestone from the purification process. Any of the commercial cells may be used as liberator cells, the blister-copper anodes being replaced by insoluble lead anodes containing 6 per cent antimony. A small amount of lubricating oil is placed on the surface of these cells to prevent acid mist from rising with the evolution of oxygen from the anode.

The total evaporation from all cells amounts to 3000 gal. daily, making ample room for the addition of water, obtained from the washing of cathodes and anode scrap and from cell cleaning and slimes washing. Attempts have been made to restrict evaporation by the use of filming oils, but these have been unsuccessful, the oil quickly disappearing and reappearing as a black greasy substance at the head of the cascades.

Solution circulation is effected by direct-driven duriron pumps in the basement.

Power

Power is supplied by two motor-generator sets which are housed in the smelter blower room close to the tank house, and controlled by the attendants in that establishment, each set comprising a 700-hp. synchronous motor taking three-phase, 50-cycle, 3000-volt alternating current, coupled direct to two direct-current, 225-kw. generators designed for a maximum delivery of 4000 amp. each at voltages varying from 5 to 90 volts.

Power requirements are high, each ton of deposited copper taking 500 to 560 kw-hr., measured in direct current. This is due to the high resistance of the electrolyte and the relatively wide spacing of the anodes and cathodes, as previously described.

There is no waste heat available from reverberatory or refining furnaces, therefore, in view of the high cost of fuel, steam for solution heating, etc., is produced by an electric boiler. This has a capacity of 700 kw., the current being fed to three electrodes at 3000 volts a.c. through a one to one ratio transformer, the latter being necessary to provide a grounded neutral. The boiler is a cylindrical vertical shell 10 ft. high and 3 ft. in diameter. Water is fed at the bottom from a 90-lb. pressure main, and a centrifugal pump circulates it from the hot well to the upper portion, where it comes in contact with the cast-iron electrodes, which are suspended in a perforated basket. The amount of steam generated is regulated by the adjustment of the pump valve, as this governs the amount of the immersion of the electrodes in the water.

Steam is generated at 75 lb. pressure and is used for heating and concentrating the solution discarded for purification, heating of cathode wash water, heating the electrolyte when necessary, and for drying slimes. The average amount of power consumed by the boiler is 300 kw-hr. per ton of copper deposited.

Handling Copper

All copper is handled and transported about the plant by means of 3-ton storage-battery mobile cranes of the wharf-loading type, the load being carried on the jib with a double chain sling. One of these machines

TABLE 1.—*Analyses of Tank-house Materials during* 1932

	Anodes	Cathodes	Slimes	Solutions		Bluestone
				Stripper	Commercial	
Cu, per cent	99.05	99.980	46.50	3.5	3.5	25.04
Ag, oz. per ton	14.75	0.320	1180.00			
Au, oz. per ton	0.43	0.008	34.00			
Sb, per cent	0.018	0.0008	1.45	0.017	0.025	
As, per cent	0.010	0.0004	0.59	0.022	0.06	
Ni, per cent	0.067	0.0009	0.05	0.22	0.80	0.15
Fe, per cent	0.015		0.09	0.10	0.22	0.12
Pb, per cent	0.230	0.0030	20.45			
Zn, per cent	0.002	Trace	0.04	0.001	0.001	
Bi, per cent	0.0002	Trace	Trace			
S, per cent	0.101	0.0050	11.83			
Se, per cent	0.031	0.0012	2.83			
Te, per cent	0.016		1.18			
CaO, per cent	0.033		0.19	0.04	0.08	
Insol, per cent	0.025		0.73			
H_2O, per cent	0.836		10.00			
Cl, per cent				0.005	0.005	
Specific gravity at 50° C				1.14	1.18	

operating on day shift only takes the anodes from the converter building to the tank house or storage dock as required and supplies the tank house from the latter when the converter plant is not operating. It also removes the cathodes to storage and loads them into railway trucks and handles anode scrap and other sundries. Concrete tracks are provided between the various points and this method of transport has proved most flexible and efficient.

Analyses

Table 1 shows analyses of the various tank-house materials during the year 1932.

Costs

The following are the average working costs of operating the tank house, the output being at the rate of 11,000 tons cathodes per annum, the number of the crew being 33, and the rate for unskilled labor being £3.30 per week of 48 hours.

Operating:	
Wages	£0.750*
Stores	0.275
Power	0.175
Maintenance:	
Wages	0.110
Stores	0.075
Transport and weighing	0.045
Assaying	0.070
Supervision	0.150
Total	£ 1.650

* All figures in this column refer to Australian currency.

Notes on Purification of Electrolytes in Copper Refining

By E. S. Bardwell* and R. J. Lapee,† Great Falls, Mont.

(New York Meeting, February, 1934)

Twenty years ago, W. T. Burns, in his paper presented at the Butte Meeting of the Institute,[1] discussed the general scheme then in use in the electrolytic copper refinery of the Anaconda Copper Mining Co. at Great Falls, Mont., for removing impurities from tank-room electrolyte and controlling its acid content. Since then marked changes have taken place in copper-refinery practice at Great Falls. The electrolytic copper refinery in use in 1913 has been replaced by a new and entirely modern plant capable of producing up to 27,000,000 lb. of cathode copper per month. With the building of the new plant in 1916, the treatment of converter anodes was discontinued. At first the substitution of refined anodes for converter anodes resulted in a decrease in the impurity content of the anodes, but as time went on, changes in metallurgical practices at the smelter resulted in substantial increases in the impurity content of the anodes. The leaching of the silver slime to remove copper before shipment to Raritan for treatment further added to the purification problem. The fact that the scheme of purifying electrolyte which Mr. Burns described in his paper has proved adequate for our needs under a wide variation of conditions speaks well for its essential soundness.

Method of Purification

Briefly, the method adopted for purifying tank-room electrolyte at Great Falls has been as follows: Each day a certain volume of tank-room electrolyte is run off and sent to the purification plant, where it is boiled down to approximately 46° Bé. The concentrated electrolyte is then sent to crystallizing tanks where the bulk of the copper is crystallized out as copper sulfate. The mother liquor is then passed through insoluble anode tanks to remove the last of the copper and most of the arsenic and antimony. It is then either returned to the tank room direct or boiled down to 55° Bé, the iron and nickel salts crystallized out and the mother liquor returned to the tank-room circuit as restored acid.

Prior to 1930 all of the evaporating was done in lead-lined tanks 13 ft. in diameter by 4.5 ft. deep, provided with lead heating coils through which steam at 30 lb. gage pressure was circulated.

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[1] W. T. Burns: *Trans.* A.I.M.E (1913) **46**, 703.

To evaporate tank-room electrolyte or silver-slime plant leach water, 1½ lb. of steam was required for each pound of water removed. To further concentrate the electrolyte after removal of arsenic and antimony in the insoluble anode tanks considerably more steam was required than for the primary evaporation. This was due in part to the fact that the boiling point of the already concentrated solution was higher than the temperature of the steam at 30 lb. gage pressure, and to the further fact that nickel and iron sludge settled out in the boiling tank and formed an insulating layer over the coils. After the charge had been boiled down and the concentrated liquor siphoned off, it was necessary to boil the sludge for several hours with water before it could be sluiced out and the boiling tank made ready for a new charge. A sludge high in arsenic and antimony gradually built up in the tanks used for primary boiling. This precipitate consisted largely of basic sulfates, which were relatively insoluble even in boiling water and had to be shoveled out from time to time. The lead heating coils, even though made of 6 per cent antimonial lead with heavy walls, tended to expand as a result of water hammer; became badly scored on the inner surface by the cutting action of water in the steam and pitted on the outer surface by galvanic action. Frequent leaks resulted, as a consequence of which solution was sometimes siphoned over into the steam lines, causing rapid corrosion of steam lines and traps.

Therefore, the boiling tank constituted a serious item of expense both from the point of operating cost and that of upkeep.

In May, 1928, natural gas became available and shortly thereafter experiments were started in developing a direct-fired evaporator. To trace step by step the experiments that resulted in the development of a satisfactory continuous direct-fired evaporator both for tank-room electrolyte and for making restored acid would require more space than is available but a few more details in connection with the problem in general may be of interest.

Purification Problems

Table 1 gives the average analysis of anodes treated during the years 1929 and 1933 inclusive. As will be noted, arsenic, antimony and tellurium are the principal impurities with which we have to contend. Inasmuch as arsenic is the only impurity that tends to build up in our electrolyte under normal conditions, it might perhaps be called the controlling impurity.

During the year 1930 the tank room at Great Falls operated at a current density of 18.3 amp. per sq. ft., a density not greatly in excess of ordinary practice in the eastern refineries. Over the four years from 1929 to and including 1932 the arsenic content of the anodes decreased while the antimony content remained constant, or nearly so. This

TABLE 1.—*Analysis of Anode Copper Treated at Great Falls*

Years	Analysis, Per Cent				
	1929	1930	1931	1932	1933 3 Months Only
Copper	99.290	99.286	99.349	99.199	99.406
Zinc	0.0008	0.0010	0.0009	0.0010	
Lead	0.015	0.020	0.012	0.008	
Arsenic	0.088	0.084	0.062	0.056	0.099
Antimony	0.072	0.073	0.075	0.072	0.062
Selenium	0.012	0.013	0.010	0.008	
Tellurium	0.084	0.073	0.089	0.069	
Nickel	0.014	0.022	0.014	0.010	
Bismuth	0.0057	0.0060	0.0054	0.008	
Iron	0.0010	0.0011	0.0013	0.0007	
Sulfur	0.0032	0.0027	0.0027	0.0024	

condition had an interesting effect upon our purification problem. During 1930 we operated the purification plant continuously in order to hold the arsenic in the electrolyte within reasonable bounds. During a part of the year 1931 and nearly all of 1932, the purification plant was operated intermittently, and then only for short periods, to control the acid content of the electrolyte. The decrease in the antimony content of the anodes and increase in the arsenic content during the early part of this year has forced us to resume operation of the purification plant.

The explanation of this phenomenon is simple and interesting. Antimony has a very limited solubility in tank-room electrolyte. Having reached the point of saturation under existing conditions, it is thrown out as a basic sulfate, taking with it basic sulfates of arsenic and probably bismuth. This is well illustrated by an experiment carried out several years ago by S. Skowronski, research engineer at the Raritan Copper Works. Skowronski electrolyzed an anode that was high in antimony but contained little or no arsenic, in an electrolyte high in arsenic but containing little or no antimony, with the result that the arsenic content of the electrolyte was progressively diminished as the experiment proceeded. The following simple experiment illustrates the same principle.

A sample of tank-room electrolyte containing 7.5 grams per liter of arsenic and 0.70 grams per liter antimony was divided into 100-c.c. portions to each of which we added antimony trioxide, Sb_2O_3, in weighed amount as shown in Table 2. Each portion was boiled for 20 min., made up to the original volume with distilled water and allowed to stand for 20 hr. At the end of this time the precipitate, which settled rapidly, was filtered off and the filtrate analyzed for arsenic and antimony.

The final result of this phenomenon, which is, as far as precipitation of arsenic is concerned, more mechanical than chemical, is that arsenic

TABLE 2.—*Arsenic Precipitation Test*

Sample	Sb_2O_3 Added, Grams	Arsenic in Filtrate, Grams per Liter	Antimony in Filtrate, Grams per Liter
Head		7.5	0.70
1	0.25	7.2	0.71
2	0.50	6.1	0.50
3	0.75	5.9	0.59
4	1.00	4.9	0.42
5	1.50	3.8	0.36
6	2.00	3.5	0.42
7	3.00	2.0	0.23

instead of tending to build up in the electrolyte is precipitated in the refining tanks and removed finally with the silver slime. Doing away, as it does, with the necessity for purifying electrolyte, this might seem at first thought to be an ideal condition, and indeed, Addicks so states.[2] However, if one takes into account the contamination of the cathode deposit and the difficulties incident to removing from solution lines the cementlike deposit of arsenic and antimony sulfates that forms whenever and wherever the temperature drops a few degrees, the prospect becomes much less inviting. Indeed, it is questionable whether it is not actually cheaper to remove the impurities from solution by purifying than to fight the various difficulties that arise by reason of the precipitation of this float slime in the tank room.

Table 3 shows the average analysis of our tank-room solutions for the years 1930 and 1932. The average analyses for these two years have been selected as showing that during 1932, when but little in the way of purification was attempted, there was no considerable building up of arsenic in the electrolyte. The lower copper content of our solutions during 1932 will also be noted. The copper content during 1932 was purposely kept low, for operating reasons which need not concern us here.

CONTINUOUS DIRECT-FIRED EVAPORATOR

Fig. 1 shows the primary evaporator used for boiling down tank-room electrolyte and silver-slime plant leach water. As a result of our experiments we found that No. 55 Ascaloy, a stainless steel containing approximately 28 per cent chromium and 0.6 per cent nickel, was well suited to withstand the corrosive action of boiling electrolyte. The evaporator consists of a series of pans arranged in cascade and forming the roof of a gas-fired furnace. Fig. 1 shows the evaporator as at present constructed. Originally the evaporator had 16 pans, each 4 ft. long by 1 ft. wide and 8 in. deep (Fig. 2*a*). These pans were made of sheet Ascaloy ⅛ in. thick with corners and spouts welded. All welds were covered for protection with a coating of Duro acidproof cement. It was necessary to protect

[2] L. Addicks: Copper Refining, 80. New York, 1921. McGraw-Hill Book Co.

the welds in this manner in order to avoid leaks brought about by intergranular corrosion due to changes in crystalline structure as a result of

TABLE 3.—*Average Analysis of Electrolyte*

Year	Specific Gravity	Analysis, Grams per Liter						
		Acid	Cu	As	Sb	Fe	Ni	Cl
1930	1.272	218	39.1	8.2	0.72	2.9	9.7	0.027
1932	1.253	213	34.5	8.7	0.60	2.5	8.0	0.028

the heat used in welding. Such corrosion as takes place is gradual. Unprotected welds will stand up ordinarily for as much as six months, but the protection described greatly prolongs their life. Mere heating of the metal to 500° to 600° C. without welding is sufficient to promote gradual corrosion. A lead-lined hood and two stacks are provided for carrying off the steam. Four atmospheric-type burners using natural gas at 5 lb. gage pressure supply the necessary heat.

Electrolyte from the tank room is pumped to a storage tank, which supplies a constant-level tank at the head of the evaporator. From the constant-level tank two 1-in. hard-lead supply lines with lead control valves supply electrolyte to the first and twelfth pans. From the bottom pan concentrated electrolyte flows via a lead-lined settling tank to crystallizing tanks. After the evaporator had once been regulated very little attention was necessary.

Table 4 gives the operating data for the first six months of operation.

TABLE 4.—*Operating Data, Primary Evaporator*

	Total Gallons	Average Gallons per Day	Grams per Liter	
			Copper	Acid
Electrolyte treated	752,030	4,345	37.5	220
Leach solution treated	136,000	786	56.2	77
Solution evaporated	888,030	5,131		
Concentrated solution produced	448,800	2,590	98.0	420
Water evaporated	439,230	2,540		

Gas used, cu. ft.	6,640,500
Cu. ft. per day	38,400
Cu. ft. per pound water evaporated	1.86

The saving in fuel over and above what would be required to boil the solution in the old steam-heated evaporating tanks amounts to about 0.9 cu. ft. of gas per pound of water evaporated.

In order to avoid the difficulties incident to welding, we have recently replaced the top twelve pans with six double-size pans made of $\frac{1}{16}$-in.

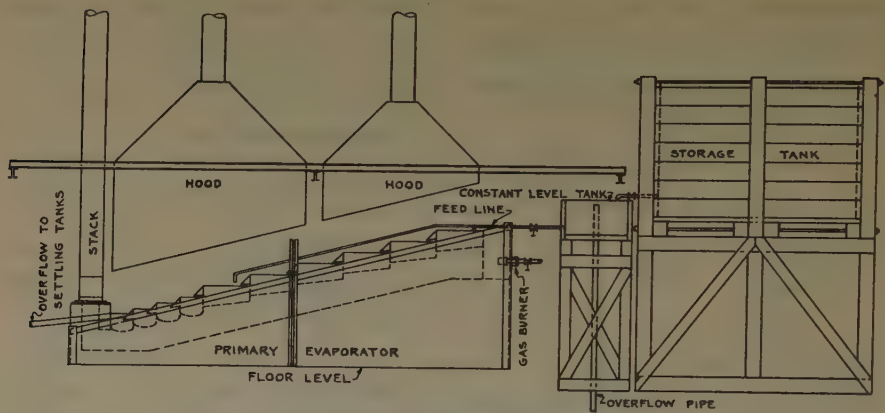

FIG. 1.—ELEVATION OF PRIMARY EVAPORATOR.

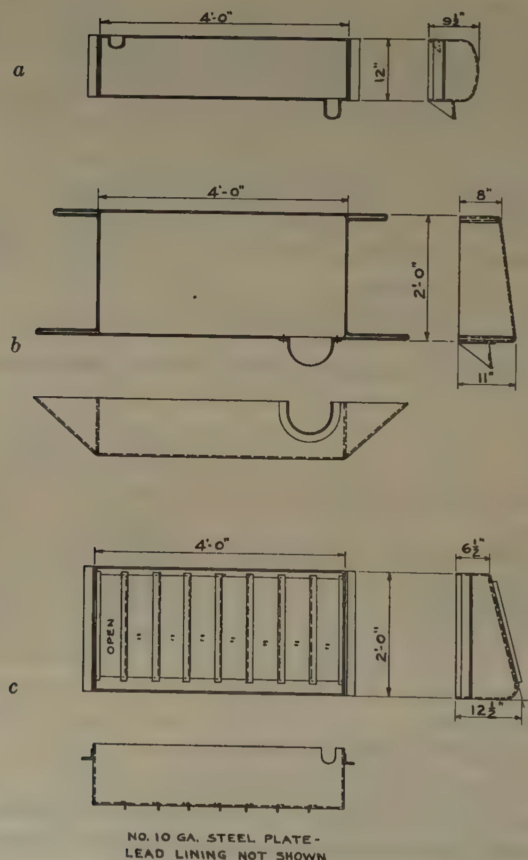

FIG. 2.—DETAILS OF PANS: (*a*) ORIGINAL ASCALOY PAN; (*b*) DOUBLE-SIZE PAN; (*c*) ONE OF TWO TOP PANS IN SECONDARY EVAPORATOR.

Ascaloy. These pans, one of which is shown in Fig. 2*b*, is bent into shape and riveted cold. Judging from our experience to date, these pans should last indefinitely, as the weld has thus far proved the only weak point.

The secondary evaporator is shown in Fig. 3. This evaporator consists of six pans arranged in cascade. The two top pans constructed as shown in Fig. 2*c* consist of iron framework lined with 8-lb. hard lead. The four bottom pans are similar to those used in the primary evaporator but are of ordinary tank steel instead of Ascaloy. Three atmospheric-type burners are used to supply the necessary heat. In this case, however, the burners are placed under the bottom pan rather than the top pan as was done in the case of the primary evaporator. Our early experiments led us to place the burners under the top pan in the primary evaporator in order to avoid having copper sulfate crystallize out in the bottom pan. As in the case of the primary evaporator the secondary evaporator is provided with a lead-lined hood for carrying away the vapor. Two lead lines bring solution from a constant-level tank to the first and the fourth and fifth pans.

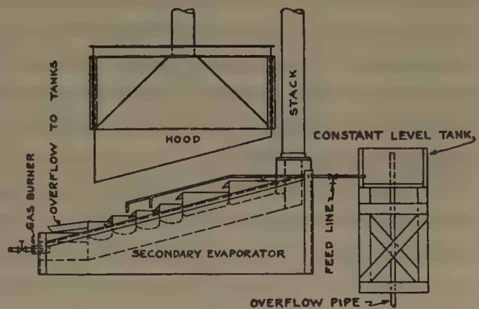

FIG. 3.—ELEVATION OF SECONDARY EVAPORATOR.

The specific gravity of the solution to be evaporated varies between 1.36 and 1.41, but such changes as occur are always gradual. We have found it necessary to keep the specific gravity of the solution leaving the two top pans about 1.44. If the specific gravity becomes much higher, scale forms in the second pan and the lead melts. On the other hand, if it is allowed to get much lower the acid attacks the first iron pan. Specific gravity readings are taken hourly and very little regulation is necessary.

Operating data showing the results obtained over the six months' period are given in Table 5.

TABLE 5.—*Operating Data, Secondary Evaporator*

	Total Gallons	Average Gallons per Day	Grams per Liter Acid
Acid treated	280,924	1,835	530
Restored acid produced	137,674	900	1,042
Water evaporated	143,250	935	

Gas used, cu. ft.	2,602,400
Cu. ft. per day	17,000
Cu. ft. per pound water evaporated	2.23

Removal of Impurities

From the primary evaporator the solution, as has already been stated, is sent to lead-lined crystallizing tanks in which the greater part of the copper is crystallized out as copper sulfate. The mother liquor is then sent to insoluble anode tanks of which there are twelve arranged in four cascades of three tanks each. These tanks are so arranged that the flow from two cascades can be joined at the end of either the first or second tanks, thus cutting out either two or four tanks should the impurity content of the solutions not require the use of all of the tanks. This arrangement permits the attainment of greater current efficiency and saving of power cost.

Table 6 shows an analysis of the mother liquor and the outlets of each of the three tanks in cascade. The solution is circulated through the tanks at the rate of six liters per minute. Antimonial lead is used as anodes and copper starting sheets as cathodes.

Table 6.—*Analysis of Solutions in Insoluble Anode Tanks*

	No. of Tanks	Specific Gravity	Grams per Liter				
			Acid	Cu	Fe	As	Sb
Mother liquor		1.425	510	14.3	4.3	18.5	1.08
Outlet No. 1 tanks	4	1.405	530	1.3	4.4	6.4	0.40
Outlet No. 2 tanks	4	1.405	545	0.4	4.4	1.8	0.14
Outlet No. 3 tanks	2	1.405	550	0.2	4.4	0.9	0.12

The bulk of the copper remaining in the mother liquor from the crystallizing tanks is deposited in the first, or head, tank. Periodically the tanks are cut out, the cathode deposit cleaned off and the accumulated sludge sluiced out, drained and shipped to the smelter. Following is an analysis of the sludge: Copper, 29.9 per cent; silver, 14.1 oz. per ton; gold, 0.13 oz. per ton; zinc, 0.03 per cent; lead, 2.7; arsenic, 31.2; antimony, 1.93; selenium, 0.004; tellurium, 1.8; nickel, 1.7; bismuth, trace; iron, 0.2; sulfur, 4.1; insoluble, 0.5. All of the silver, gold, selenium and tellurium come from the leaching of silver slime, as these elements do not exist in tank-room solutions.

If, as has been previously stated, our tank-room solutions are not too high in iron and nickel, the solution after being depleted in arsenic and antimony is returned to the tank room. If it is desired to lower the iron and nickel content of the tank-room solutions, this solution is sent via the secondary evaporator to crystallizing tanks where the iron and nickel are removed in a sludge containing about 12.8 per cent nickel and 3.9 per cent iron and the mother liquor is returned to the tank room as restored acid.

At present the nickel-iron sludge is being wasted as the recovery of its nickel content is not profitable. The analysis of the sludge shown above is quite favorable for the recovery of nickel; very much more so, in fact, than formerly was the case. Inasmuch as the nickel sludge contains 25 per cent sulfuric acid, the usual method of neutralizing with milk of lime is out of the question. The following tentative scheme has been worked out for the treatment of this sludge.

Inasmuch as ferrous sulfate decomposes at 550° C. and nickel sulfate at 703° C., it was thought that by roasting at an intermediate temperature it should be possible to render the iron insoluble while leaving the nickel sulfate water-soluble. We found that by roasting for 2 hr. at 650° C. 99 per cent of the nickel was water-soluble while only 14.0 per cent of the iron remained water-soluble. The suggested scheme then, if nickel salts were to be made from this material, would be to roast under carefully controlled temperature condition at 650° C., leach with boiling water, precipitate the iron with milk of lime and filter. The iron-free nickel sulfate solution could then be evaporated and the nickel salts crystallized out.

TABLE 7.—*Disposition of Impurities*

	Arsenic, Lb.	Per Cent of Total	Antimony, Lb.	Per Cent of Total	Nickel, Lb.	Per Cent of Total
Year, 1932						
Liberated from anodes	62,185		79,952		11,104	
Increase in electrolyte	2,392		720[a]		5,949	
Total removed	59,793		80,672		5,155	
Distribution of impurities:						
To silver slime	36,168	60.5	56,388	69.8	470	9.1
To cathodes	1,880	3.1	1,990	2.5	1,548	30.0
Removed in purifying	21,745	36.4	22,316	27.7	3,137	60.9
Volume of electrolyte purified: 1,274,644 liters.						
Year, 1930						
Liberated from anodes	187,216		163,292		48,624	
Increase in electrolyte	5,157[a]		469[a]		7,633	
Total removed	192,373		163,761		40,991	
Distribution of impurities:						
To silver slime	60,914	31.6	118,505	72.4	1,772	4.3
To cathodes	3,367	1.7	3,051	1.8	4,174	10.2
Removed in purifying	128,092	66.7	42,205	25.8	35,045	85.5
Volume of electrolyte purified: 9,926,073 liters.						

[a] Decrease.

It has already been shown that where the antimony content of the anodes is higher than the arsenic content, the electrolyte does not tend to build up in arsenic to as great an extent as where the reverse is true. In order to bring out this fact more clearly, Table 7 has been prepared, showing the distribution of the portion of the impurities coming in that was eliminated by purifying solution or in cathode copper and silver slime. In studying these data it should be borne in mind that the tonnage treated in 1930 was approximately double that treated in 1932. The extent to which purification of electrolyte was attempted during the two periods selected for comparison will be clear if we say that during 1930 the volume of electrolyte treated through the purification plant was nearly 2.7 times the total volume in circulation at any one time, while during 1932 the volume treated was approximately one-third the total volume.

During 1932, when little purification was attempted, nearly twice the percentage of total arsenic was thrown into the silver slime and the percentage of total arsenic, antimony and nickel trapped in the cathode deposit likewise increased. The bulk of the antimony goes into the silver slime in either case.

Regardless of the current density at which a tank room may be operated, antimony is one of the most troublesome impurities with which the refiner has to contend.

DISCUSSION

O. Nielsen, Ilsenburg am Harz, Germany (written discussion).—The problem of evaporating electrolytes by burning fuel only has already been solved and applied at the electrolytic plant built by the writer at Kupferwerk Ilsenburg. The concentrating of the electrolyte is performed on a shallow mold made of ordinary soft iron boiler sheet under which a grate for burning coal is arranged. As soft iron is rapidly attacked by the diluted acid in the electrolyte, the method was feasible only by using this trick:

As acid of about 42° Bé. does not attack soft iron, care is taken never to let fall the specific weight of the content of the pan lower than to 42° Bé. When the pan once is filled with solution of this gravity, evaporation is started and continued up to 54° Bé. (measured as hot). By this a suspension of salts and concentrated sulfuric acid has been formed. About half the content is withdrawn to a settling tank where the salts and the acid separate. The acid is taken back to the main electrolyte. To the remainder of the pan content so much fresh electrolyte is added as to lower the specific gravity to 42° Bé. The evaporation starts again and is continued to 54° Bé. a.s.o.

The iron pan, which costs little, lasted in the beginning for about three weeks; later on, when the man in charge of it had become more familiar with his work, for about nine months, and now we have nearly forgotten when the last pan was installed.

It would be of general interest, if Messrs. Bardwell and Lapee could give further information as to the equilibrium of copper in the electrolyte, and whether the copper sulfate produced by the first concentrating of the electrolyte is sent back to the electrolyte after resolution.

Recovery of Precious and Secondary Metals from Electrolytic Copper Refining

By M. A. Mosher,* Perth Amboy, N. J.

(New York Meeting, February, 1934)

Most modern electrolytic copper refineries have found it expedient to build and operate a complete secondaries plant for the treatment of the electrolytic slime accumulated in the cells during the corrosion of the copper anodes. In the earlier years this was done for the sole purpose of promptly recovering in marketable form the silver and gold contained in the slime. Later, improvements in refining methods made possible the recovery of the more precious metals such as platinum, palladium, osmium and iridium, and in recent years, due largely to an increasing market, many secondaries plants have undertaken the commercial production of selenium and tellurium. The complete up-to-date secondaries plant, or silver refinery as it is usually called, is therefore equipped for the production in marketable form of from six to eight metals. The physical plant and operations may be separated into six main divisions:

1. Slime treatment, in which the raw slime is treated by wet methods for the removal of its high copper content and portions of other impurities such as arsenic, selenium and tellurium.
2. Furnace refining, in which the remainder of the base-metal content of the slime is removed by fluxing and fire-refining.
3. Electrolytic parting and refining, in which the silver and gold are parted and obtained in the pure state.
4. Melting of silver and gold into marketable form.
5. Recovery and refining of the more precious metals.
6. Recovery and refining of selenium and tellurium.

The Silver Refinery of the Raritan Copper Works at Perth Amboy, N. J., is a complete secondaries plant and the following description of its equipment and processes is, in a general way, typical of other plants of its kind.

Slime Treatment

The raw slime, pumped with a minimum amount of cell solution from the electrolytic copper refinery, is discharged into the center of circular settling tanks 15 ft. in diameter by 8 ft. deep, where the slime settles out and the solution overflows peripherally to return to the copper refinery by gravity. The slime is then dropped through a bottom outlet to a

* Silver Refinery Superintendent, Raritan Copper Works.

mixing tank where it is kept at the proper consistency, about 30 per cent solids, by mechanical stirring, and fed to an Oliver continuous filter.

This filter, which is mainly of hard lead construction and completely proof against sulfuric acid-copper sulfate solution, has a drum 3 ft. in

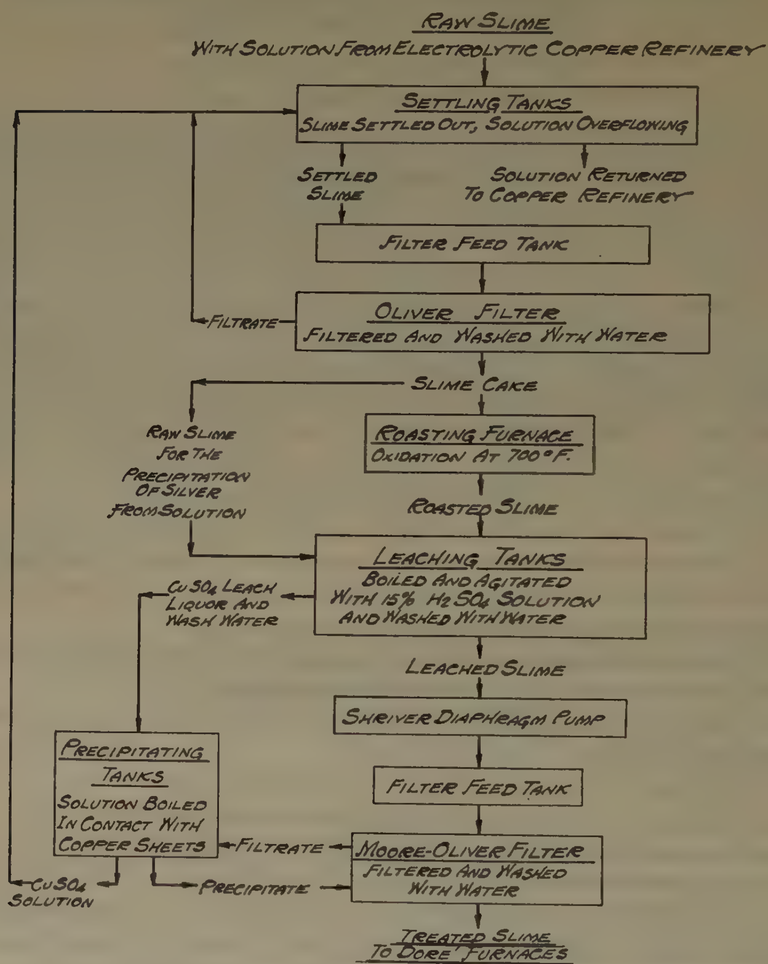

FIG. 1.—FLOW SHEET OF SLIME TREATMENT IN ELECTROLYTIC COPPER-REFINERY SECONDARIES PLANT.

diameter with a 4-ft. face. The cake, which attains a thickness of ½ in., is washed with water from a spray pipe as the drum revolves, and discharged with a moisture content of less than 35 per cent into lead-lined trucks.

The slime cake is then taken to a pan roasting furnace 27 ft. long by 9 ft. wide, transferred to sheet-iron pans 48 by 22 by 4 in., which are then stacked in sets of three. One of these sets is introduced on rollers

into each of the 10 doors opening on either side of the furnace. Heating is obtained by means of two oil burners at one end of the furnace, the pans being protected from the flame by the brick roof over the divided firebox, which extends the full length of the furnace under either row of pan racks. At the rear end of the furnace the hot gases are conducted up and back over the pans to the gas stack at the burner end. The slime is left undisturbed under a temperature of about 700° F. for a period of 8 hr., ample air being admitted to the combustion chambers to insure an oxidizing atmosphere. The pans are then removed from the furnace and elevated by means of an electric hoist to the leaching-tank platform where the entire roaster charge of approximately 3000 lb. of oxidized slime is dumped into one leaching tank containing a boiling 15 per cent sulfuric acid solution.

These tanks, circular in shape and 5 ft. in diameter by 6 ft. deep, consist of a channel-iron skeleton lined with 10-lb. hard lead, and stirring is obtained with a slow-moving two-bladed cast-lead paddle, gear-driven from above. Agitation is continued for several hours until the copper is leached out and a test of the liquor shows silver in solution. A quantity of raw slime, having sufficient metallic copper content to precipitate the silver from solution, is then added and after a short period of settling the leach liquor is siphoned off. Water is run into the tank and boiling and agitation resumed in order to wash the entrained leach liquor out of the slime. Five washings are usually sufficient and the slime is then dropped through a bottom outlet into a sump from which it is pumped by a Shriver diaphragm pump to the mixing tank, preparatory to dewatering on a Moore Oliver-type filter. The cake is given a final wash on this filter and discharged into lead-lined trucks ready for furnace refining.

Any leach liquors or wash waters that upon test show selenium or tellurium in solution are pumped to precipitating tanks, where these elements are removed by boiling in contact with copper. The precipitate thus formed is later treated in the doré furnaces for the recovery of any values it may contain.

All boiling of solutions is done with steam introduced through open-end lead pipes or coils, dilution from condensation being of no importance. Leach liquors and wash waters are removed from the leaching tanks by means of large gravity siphons into a sump tank from which they are pumped wherever desired with a 2-in. vertical centrifugal pump. All solutions free from selenium and tellurium are pumped back to the settling tanks, from which they return to the electrolytic copper refinery by gravity. Solution lines are all of hard lead and fitted with lubricating duriron plug cocks.

The standard dry vacuum system is used for both Oliver-type filters, each being equipped with a 9½ by 8-in. vacuum pump and a 2-in. Olivite centrifugal pump to remove the filtrate against the vacuum.

Typical analyses of the electrolytic slime before and after this treatment are given in Table 1.

TABLE 1.—*Typical Analyses of Electrolytic Slime*

	Raw Slime	Treated Slime
Silver, oz. per ton	5130.88	7684.00
Gold, oz. per ton	101.52	146.20
Copper, per cent	18.10	1.74
Lead, per cent	3.19	7.18
Arsenic, per cent	5.63	3.60
Antimony, per cent	4.66	13.97
Selenium, per cent	5.31	4.51
Tellurium, per cent	6.14	3.49

The percentage difference unaccounted for in each analysis is made up of sulfates, oxides and silica.

FURNACE-REFINING OF SLIME

The treated slime, containing less than 2 per cent copper and from 25 to 30 per cent moisture, is charged into a small reverberatory furnace and melted down, a complete charge consisting of 8 to 10 tons of wet slime introduced into the furnace one ton at a time over a period of approximately 20 hr. When completely melted, a vitreous slag, consisting of the lead and a large portion of the antimony combined with the silica content of the slime, rises to the surface and is skimmed off. Two air pipes are then introduced into the melt, and working of the charge with rabbles is continued for about six hours until the remainder of the antimony has been volatilized. The arsenic and a part of the selenium and tellurium are also driven off during this period. Several hundred pounds of fused sodium carbonate is next introduced to break up the matte, after which the removal of the remaining selenium, tellurium and other base metals in the form of a water-soluble slag is carried on with the use of sodium nitrate as a flux and oxidizing agent. As this slag forms in sufficient quantity it is skimmed off and more niter worked in, this procedure being continued until the bath of doré (silver and gold) remaining in the furnace is of a high degree of purity, averaging 980 fineness silver and 15 fineness gold. The total time of the furnace refining cycle is from 48 to 72 hr., depending on the base-metal content of the slime.

This silver-gold metal is then ladled out into water-cooled, double-face, reversible molds to produce anodes for the subsequent electrolytic parting. Two types of anodes are cast, one 19¼ by 21½ by ½ in. for the Thum electrolytic process, and the other 6¼ by 9 by ½ in., with a small lug at one end, for the Moebius process.

The furnaces have a hearth area 6 by 8 ft. The bottom and sides are lined with magnesite brick and the roof is of clay brick. They are oil-fired with a single Quigley-type burner through a combustion chamber at the rear, the oil consumption being from 15 to 20 gal. of light fuel oil per hour. The working of the charge is done through a front door above which the uptake carries the gases into the main flue, while the charging of slime is done through a door midway in the side.

A motor-driven draft fan 6 ft. in diameter midway in the main flue system furnishes the draft for the furnaces and keeps the gases moving through the entire flue system. Ample cooling area and chambers for settling out the dust are provided before the gases enter the base of the first of three steel-supported lead towers 25 ft. high by 4 ft. 6 in. in diameter. As the gases rise they are scrubbed and moistened with water falling from the top of the tower. This treatment is repeated in the downward passage of the gases through a second tower, and in rising in the third tower they drop their excess moisture before entering a Cottrell electrical precipitator.

The lower ends of the three lead towers in the scrubber system are immersed in a water-seal tank from which the scrubber water overflows into a settling tank. The clarified discharge from this settler is conducted to a 3-in. duriron centrifugal pump of capacity ample to deliver the water again at the top of the first two towers into reservoirs through the perforated bottoms of which it falls as a spray through the columns of gas. Adequate scrubbing and moistening is obtained in this manner without the use of baffles or tower packing.

For final treatment the gases are passed through a standard tube-type three-unit Cottrell electrical precipitator. Each unit consists of 26 vertical lead tubes 8 in. in diameter by 12 ft. long equipped with a carefully centered lead-covered electrode wire on which the high-tension electric current is impressed. The four corner tubes of the unit are 11 in. in diameter and have a correspondingly larger lead-covered iron-rod electrode sufficiently strong to carry an insulated grid, which keeps the other wire electrodes centered in the 8-in. tubes. Even distribution of the gas is obtained in a chamber beneath the tubes while another chamber over the top of the tubes protects the process from the weather, at the same time providing for the exit of the gases through a vertical stack in which tests of the efficiency of precipitation can be made. One of these units ordinarily is capable of clearing the gases from the operation of two doré furnaces for a period of 12 to 16 hr. The high-tension unidirectional current for the precipitation is obtained by transforming 220-volt alternating current to 50,000 volts and then rectifying mechanically.

Operation of the three units is so arranged that there is always available a clean unit that may be put into service while either or both of the other units are shut down for cleaning. Every approach to any unit in

operation is guarded by one of a system of Robinson interlocks which enables operators or repairmen to work on adjacent units without fear of injury from the high-voltage current.

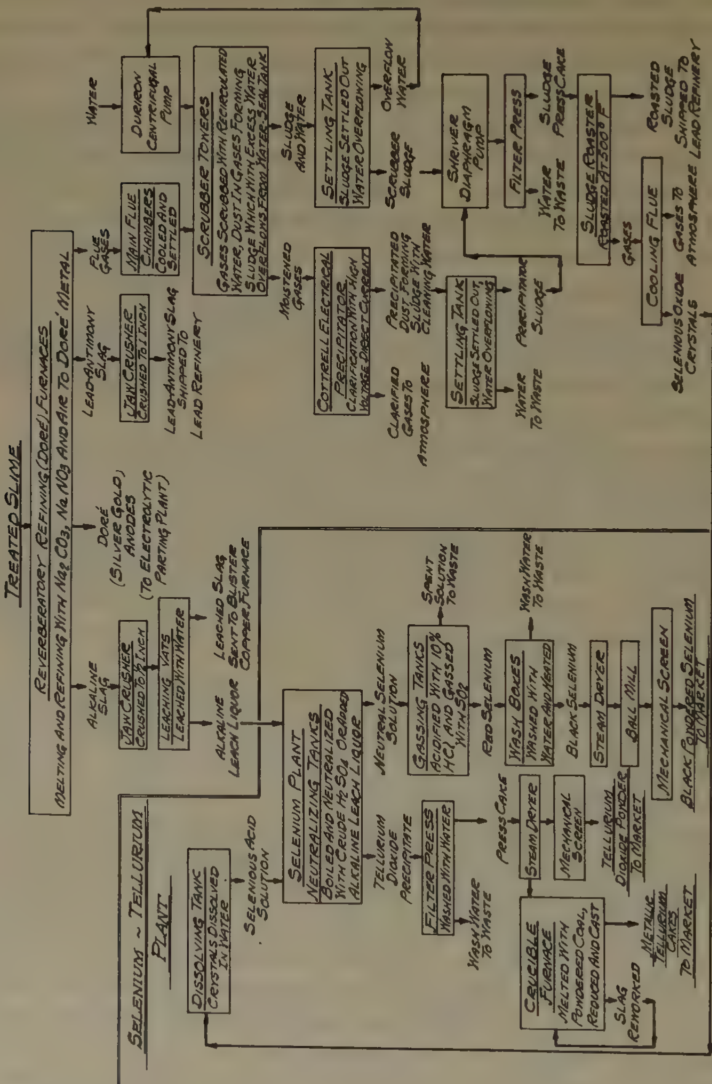

FIG. 2.—FLOW SHEET OF FURNACE REFINING AND SELENIUM-TELLURIUM RECOVERY AT ELECTROLYTIC COPPER-REFINERY SECONDARY PLANT.

The precipitate of sludge accumulated on the walls of the tubes and electrodes is daily washed down by hand with water from a high-pressure hose and sluiced to a settling tank. This sludge, together with that accumulated in the settlers of the scrubber system, is filter-pressed with a Shriver diaphragm pump and plate and frame-press installation.

The sludge-press cake is then delivered to a reverberatory roaster, where it is given a slow low-temperature roast for the removal of selenium and arsenic. In the flue chambers taking the gases from this roaster the selenium is deposited as selenious oxide crystals, which are periodically removed to the selenium plant, while the arsenic passes out into the air through a high stack. The material remaining on the furnace hearth consists mainly of lead and antimony oxides together with the silver and gold values contained in the original sludges. This roasted sludge is packed in bags and shipped to a lead refinery for final treatment.

The vitreous lead-antimony slag skimmed off early in the furnace-refining of the treated slime is reduced in a jaw crusher and also shipped to a lead refinery for final treatment.

In order to insure the subsequent removal from the cycle and the recovery of the selenium and tellurium, every precaution is taken during the fluxing stage of the furnace refining operation to make the slag as completely water-soluble as possible by means of the proper use of both sodium carbonate and sodium nitrate. This alkaline slag is reduced to ½ in. in the jaw crusher and dumped into concrete leaching vats 15 by 15 ft., which are equipped with burlap and sand filter bottoms. Ten to fifteen 24-hr. leaches with water remove the selenium and tellurium contents in the form of a highly caustic solution of sodium selenite and sodium tellurite which is pumped from beneath the filter bottoms direct to the selenium department for treatment. The insoluble slag residue remaining after leaching is dried and returned to the blister-copper furnaces.

Electrolytic Parting and Refining of Silver and Gold

The standard processes employed for the electrolysis of silver-gold anodes do not lend themselves to the elimination of any appreciable amount of base-metal impurities, since the latter may be rendered soluble as the anode corrodes and then be plated or cemented out at the cathode with the silver. The anode metal is brought to the highest possible purity therefore in the doré furnace, 995+ parts of silver plus gold, and the electrolytic treatment becomes simply a parting process to separate the silver from the gold and any other precious metals the anode may contain. Two types of electrolytic equipment are commonly employed, the Thum or horizontal process, and the Moebius or vertical process, each having its advantages and disadvantages, which will be discussed later.

In the Thum process the anode is placed within a wooden frame lined with muslin and covered with a close-fitting canvas sack. This frame in turn is placed in a heavier frame which, resting on the sides of the electrolytic cell, is partly immersed in the electrolyte and equipped with slats across the bottom to support the weight of the anode. This anode frame assembly covers about one-half the solution area of the cell, leaving

the other half open. The cells themselves, measuring 48 by 24 by 7 in., were formerly of vitrified chemical stoneware or porcelain but are now being replaced with asphalt-lined concrete tanks. The bottoms of these tanks, which must serve as cathode surfaces, are lined with slabs of graphite and carbon ½ in. thick. The current enters the cell through a

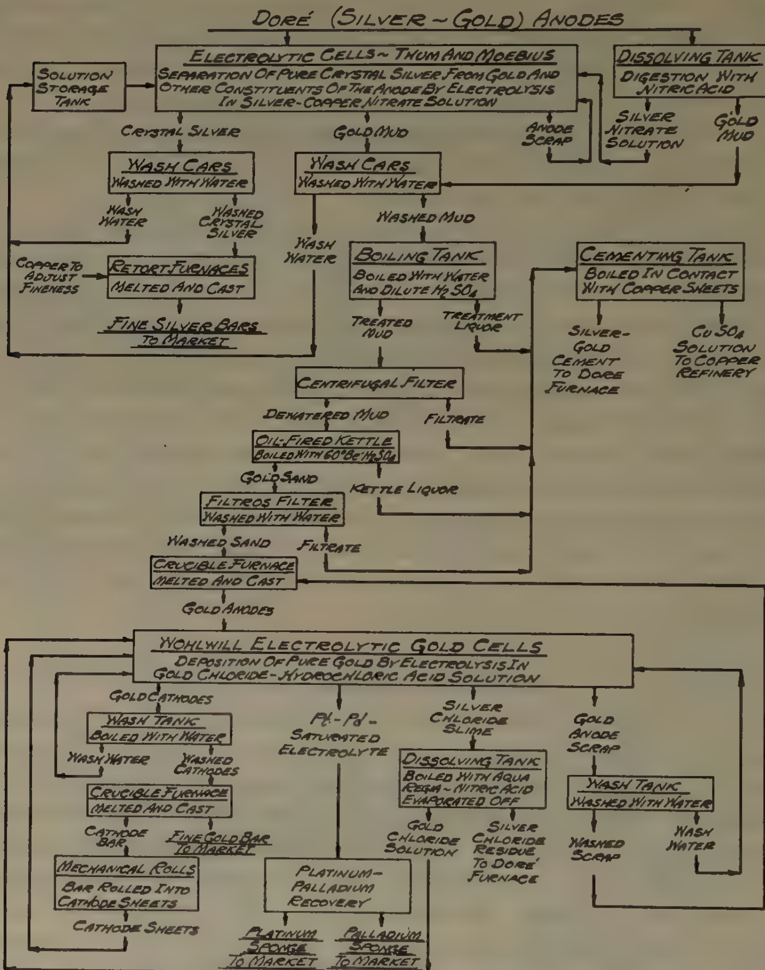

FIG. 3.—FLOW SHEET OF ELECTROLYTIC PARTING AND REFINING OF PRECIOUS METALS IN ELECTROLYTIC COPPER-REFINERY SECONDARIES PLANT.

silver contact piece which rests on the submerged anode in the tray, and is conducted from the cell through another silver contact which rests on the graphite slabs underneath the solution at the open end of the cell. The silver corroded from the anode passes through the muslin and canvas diaphragms, deposits on the carbon-slab bottom underneath the tray, and is scraped forward periodically by hand to the open end of the tank to

make way for continued deposition. The accumulated deposit is removed from the open end of the tank to a filter-bottom wash car once each shift. As the anode corrodes away, another is placed on top of it so that each anode in turn is completely electrolyzed, leaving no scrap to be remelted.

The anode current density is 50 amp. per sq. ft. with a maximum current per cell of 150 amp. at 3 to 3½ volts, the cells being arranged in series in sections of 21 tanks. The electrolyte is a neutral silver-copper nitrate solution analyzing 60 grams per liter silver and 40 grams per liter copper.

In the Moebius or vertical electrolytic parting process the same solution and current density are employed, but the anodes and cathodes are suspended vertically in a cell measuring 24 by 29 by 22 in. The anodes, 9 by 6¼ by ½ in. with a lug for hanging, are suspended in rows of three in canvas bags, the anode bar across the tank carrying the sack and its frame as well as the hangers supporting the anodes themselves. Cathode sheets of chromium iron 12 by 20 by 1/16 in. are suspended from the busbars at the sides of the tank. The cathode surfaces are continually scraped free of the crystal silver deposit by wooden blades attached to an oscillating rack which rests on the framework over the tanks and is mechanically operated. The crystal silver scraped from the cathode sheets falls through the solution into a wooden tray, which is suspended from the electrode assembly frame over the tanks. As the anodes corrode away new ones are hung in their places and the scrap lugs, which otherwise would have to be remelted, are placed in the Thum anode trays for complete corrosion.

Each tank contains five anode groups and six cathode sheets, connected in parallel. The cells are connected in series in sections of six with a maximum current density of 450 amp. at 2½ to 3 volts.

After a full producing day the entire operating equipment of a section is mechanically raised as a unit to a position 1 ft. above the cells. The trays, which have hinged bottoms, are then emptied of the accumulated crystal silver deposit into filter-bottom wash cars.

The advantages of the Thum process are: (1) Mechanical and current conductor simplicity, which make for low cost of installation and maintenance; (2) complete corrosion of the anodes, which may be of any size, shape, or weight within the capacity of the anode tray; (3) no necessity for shutdown while the crystal silver production is being removed.

The disadvantages are: (1) Corrosion from but one face of the anode, resulting in low production per unit of floor space and volume of electrolyte; (2) dependence on and cost of hand labor for scraping the deposit from the cathode; (3) lack of regular stirring or circulation of electrolyte in the cell.

For the Moebius process the advantages are: (1) Low labor cost due to mechanical scraping of the deposit from the cathode, with the added benefits of regular stirring of the electrolyte, a more uniform deposit, and a current efficiency of 98 per cent, which is 5 to 10 per cent higher than that obtained in the Thum process; (2) close spacing of electrodes, together with corrosion of both faces of the anodes, resulting in high production per unit of floor space and volume of electrolyte; (3) no necessity for muslin diaphragms, because the anode does not rest on the accumulated gold mud.

Its disadvantages are: (1) Mechanical and current conductor complexity, resulting in a high installation and maintenance cost; (2) complete shutdown necessary for the removal of the crystal silver production.

In both the Thum and Moebius processes the gold accumulates around the anodes in the sacks as a black slime or mud contaminated with particles of silver and basic copper salts. This mud is removed from the Thum trays after every three days of operation by taking out the muslin tray lining and washing it into filter-bottom wash cars. In the Moebius system the anode sacks are removed from the assembly and washed out after the same period. The solution is filtered off the mud in the wash cars and the mud transferred to a wooden boiling tank 3 ft. in diameter by 3 ft. deep, where it is boiled up with water to which is then added a small quantity of sulfuric acid to break up the basic copper salts. After settling has taken place the wash liquor is siphoned off and the mud is transferred to a Tolhurst centrifugal filter to be finally washed with water and dried. The dewatered mud is charged into an oil-fired cast-iron kettle 4 ft. in diameter, where it is boiled for about 3 hr. in 60° Bé. sulfuric acid. This boiling removes the copper and most of the silver from the mud and leaves a residue of brown sand which assays about 970 parts gold. This sand is thoroughly washed on a Filtros-block filter ready for melting down for the production of gold anodes.

Silver-copper nitrate wash waters from the washing of crystal silver and the preliminary washing of the gold mud are filtered and elevated by means of a small hard-rubber pump to a chemical stoneware storage tank from which hard-rubber distribution lines run to convenient sections of the parting plant. This wash water is used to make up electrolyte evaporation and withdrawals resulting from the removal of crystal silver and gold mud. Sufficient electrolyte is entrained in the latter and destroyed in its subsequent treatment to obviate the necessity of special withdrawals to maintain the purity of the electrolyte. Silver nitrate solution for replenishing the electrolyte is made by digesting doré anodes in nitric acid.

Gold mud wash water containing sulfuric acid and the strongly acid boiling kettle liquors, as well as the water resulting from the washing of the gold sand, are collected in a sump tank and pumped to a special

cementing tank in the slime plant, where the precious metals are precipitated by boiling the solution in contact with copper sheets. The gold-silver cement thus produced is periodically refined and cast into anodes in the doré furnace.

Melting and Casting of Silver

The crystal silver, after being thoroughly washed with water in the wash cars, is charged into clay-graphite retorts and melted. These bottle-shaped retorts, 38 in. high, are mounted in oil-fired tilting furnaces of the Faber-du-Faur type and hold a charge of about 30,000 troy ounces. The consumption of light fuel oil with a small Quigley-type burner is 8 to 10 gal. per hour. The charging and melting of a full charge requires 5 to 6 hr., after which the molds are brought up on a carriage and the standard-size bars are poured by tilting the furnace, about 45 min. being required for the casting of the usual 25,000-oz. lot in 1000-oz. bars. The fineness of the silver is adjusted to 999, the standard for the fine silver market, by adding to the melt, just before pouring, a few ounces of pure copper.

Electrolytic Refining of Gold (Wohlwill Process)

The gold sand is melted in No. 25 graphite crucibles in a cylindrical oil-fired crucible furnace, the Quigley-type burner consuming about 8 gal. of light fuel oil per hour. The metal is cast into vertical closed molds to produce gold anodes averaging 980 fineness and 140 troy ounces in weight, the dimensions being 9 by 5 by 3⁄8 in. The electrolysis of these anodes is accomplished by the Wohlwill process, in which from six to eight of them are suspended in pairs from silver anode bars in the electrolyte in a small stoneware electrolytic cell measuring 20 by 12 by 12 in. Cathode starting sheets of fine gold rolled to a thickness of 0.003 in. and having an immersed area 12 by 2 in., are suspended in rows of four from silver cathode rods spaced between the anode bars. A current of 450 amp. is put through the cell, giving a voltage of 1½ volts and an anode current density of 125 amp. per square foot with six anodes. The electrolyte is a gold chloride solution containing from 80 to 100 grams per liter of gold and 10 per cent free hydrochloric acid. Heat to maintain the minimum operating temperature of 140° F. is obtained by means of a steam coil placed around the stoneware cell and enclosed by a lead-lined wooden box, sand being packed between the lead and the stoneware cell to give even distribution of the heat from the coil. Circulation of the electrolyte during operation is maintained by means of a Pohlé air-lift. The anodes are corroded to 5 per cent scrap in from 16 to 18 hr., the pure gold plating out on the cathode sheets in a rough, somewhat nodularized, crystalline deposit which adheres firmly.

The cathodes are removed from the cell, thoroughly washed, dried and melted down in a clean crucible from which the gold is cast into an open mold to produce a bar weighing from 700 to 800 oz. and assaying 999.7+ parts per thousand. The anode scrap is washed, dried and remelted with the next batch of gold sand.

Silver, the chief impurity in the gold anodes, forms silver chloride during the electrolysis and falls to the bottom of the cell as a slime together with varying amounts of gold. This slime is removed periodically, leached with aqua regia for the removal of gold and the production thereby of gold chloride solution to be used in replenishing the electrolyte, the gold content of which is gradually depleted during electrolysis. The silver chloride remaining after this leach is returned to the doré furnace.

Strong wash waters are used to take up electrolyte evaporation, while dilute waters are sent to the cementing tank. Auxiliary equipment used in this process consists of two electric drying ovens and power-driven 6-in. rolls for the rolling down of a small bar of fine gold into cathode starting sheets when needed.

Fine silver bars are first weighed on a springless scale and finally weighed with careful checking over a bullion balance of 3000-oz. capacity and 0.005-oz. sensitivity. Fine gold is weighed in the same careful manner.

Platinum and Palladium, Selenium and Tellurium Recovery and Refining

Any platinum and palladium present in the original raw slime is finally concentrated in the gold anodes and during their electrolysis goes into solution in the electrolyte, the platinum and palladium content of which can be permitted to go as high as 60 grams per liter. The electrolyte is then removed, oxidized with nitric acid, and the combined salts of platinum and palladium precipitated by the addition of ammonium chloride. Separation and eventual refining of the platinum and palladium are carried out by the usual chemical methods, both metals being produced in the sponge form and of 99.5+ per cent purity.

The raw material for the selenium and tellurium plant is the caustic solution obtained from the leaching of the soda-niter doré-furnace slag, together with a water solution of the selenious acid crystals collected in the flue of the sludge roaster. This combined solution is brought to the point of neutrality while boiling in two lead-lined tanks, 10 by 6 by 6 ft., either by the addition of crude sulfuric acid or additional slag liquor, and complete precipitation of the tellurium as tellurium dioxide is thereby obtained. The precipitate is allowed to settle and the neutral solution siphoned off into one of the two large wooden gassing tanks, 10 by 6 by 8 ft., where it is acidified with 10 per cent of hydrochloric acid and 5 per cent of sulfuric acid by volume and gassed with sulfur dioxide

generated in a horizontal water-cooled sulfur burner. Complete precipitation of the selenium in the red amorphous form is obtained with from 24 to 72 hr. of gassing. After settling has taken place the solution is siphoned off to waste and the selenium dropped through a bottom outlet to a wash box having a burlap filter bottom, where it is washed with water. The washed red selenium is converted to the black form by the application of steam and then transferred to pans in which it is dried in a steam oven. The dried product is ground in a ball mill for several hours and mechanically screened through 100 mesh for the market. This powdered selenium, analyzing 99.5+ per cent, is packed in containers for shipment or, if required in vitreous form, is melted in iron pots and cast into cakes or sticks.

For the recovery of tellurium the tellurium dioxide precipitate from the neutralizing tanks is pumped to a filter press by a Shriver diaphragm pump. The cake is thoroughly washed and then transferred to pans and dried in a steam oven. If it is to be marketed as tellurium dioxide, it is screened through 100 mesh and packed for shipment. To produce metallic tellurium the dioxide is mixed with powdered coal and reduced in a clay-lined graphite crucible mounted in a Monarch oil-fired tilting furnace. The metal thus formed is then poured into cakes or sticks. If powdered tellurium is required, the metallic cakes, which are brittle and show a pronounced crystal fracture, are broken up and ground in a ball mill. The product is screened to obtain the desired grade of powder and packed in containers for shipment.

Summary of Major Equipment and Plant Capacities

The slime plant, with 10 slime-leaching tanks, can handle 120 dry tons of raw slime per month and turn out from 60 to 90 dry tons of treated slime per month, depending on the percentage of silver it contains.

The furnace plant, with two of the three furnaces for the refining of slime to doré in continuous operation, can refine 150 dry tons of treated slime per month, there being ample capacity to handle, in addition to the output of the Raritan slime plant, any treated slime or crude doré bullion received from other plants of the Anaconda Copper Mining Co. The capacity production of two furnaces working on slime varies, according to the silver content of the slime, from 1,250,000 to 2,000,000 oz. of doré anodes per month.

The capacity of the parting plant, with the three 6-cell Moebius sections and six 21-cell Thum sections in operation, is 2,250,000 oz. of crystal silver per month. The melting of this amount and the casting into fine silver bars can be taken care of with one of the two retort melting furnaces in continuous operation.

Facilities for refining gold mud are ample to handle an amount containing 35,000 oz. of gold per month. The Wohlwill electrolytic plant

with one of its two cells in operation will produce 25,000 oz. per month, so that its capacity is greater than that required to refine the gold resulting from full operation of the parting plant on doré running as high as 15 parts per thousand gold. Platinum and palladium refining facilities are adequate for the production of 200 oz. per month of the two metals. The selenium plant is capable of a production of 3000 lb. of powdered selenium per month and the tellurium plant can turn out 1500 lb. of metallic tellurium per month.

Development and Use of Anaconda Electro-sheet Copper

By William M. Shakespeare,* Perth Amboy, N. J.

(New York Meeting, February, 1934)

Thin sheet copper in wide widths and long lengths is being made electrolytically on a commercial scale at the Raritan Copper Works, Perth Amboy, N. J. This achievement is the result of three years of intensive development preceded by almost ten years of intermittent research.

While the idea of producing sheet copper by the electrolytic process is old, the literature on the subject is limited. It is evident, however, that the idea has been attractive to many investigators for years, if the issued patents in the field may be taken as an indication. These patents may be divided into three groups. In one group metal is deposited on some form of endless cathode, of which a revolving cylinder or drum is the most common. The cathode moves slowly and at such a rate that metal of the desired thickness is deposited during one pass through the electrolyte. The deposited metal is stripped off the cathode at a point above the solution level and, after washing and drying, is wound up immediately. In the next group of patents a band of thin metal is increased in thickness by being guided through an electrolytic tank in opposition to suitable anodes. The third group covers means of producing sheets of definite dimensions where flat plates or revolving cylinders are used as cathodes. With such cathodes, sheets of definite dimensions are built up to the desired thickness and then stripped off.

The process as worked out at the Raritan Copper Works combines the methods of the first two groups. In the first stage a sheet weighing 1 oz. to the square foot is produced on a revolving cylinder. This sheet is increased to any desired thickness in the second stage.

In the first stage (Fig. 1) a lead-covered copper drum is mounted horizontally on a lead-lined concrete tank so that it is partly immersed in the electrolyte. The cylindrical surface of the drum immersed in the electrolyte is opposed by lead anodes spaced about ½ in. from it. The electrolyte, made up of appropriate quantities of copper and sulfuric acid, flows into and out of the tank continuously. The ebullition of gas at the surface of the anodes induces a vigorous circulation of electrolyte between the anodes and the drum surface, which is augmented by means of compressed air introduced through a pipe between the ends of the anodes under the center of the drum, not indicated on the diagram. This

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vigorous circulation is necessary in order that copper may be deposited at a comparatively high rate. The deposited sheet is continuously removed from the surface of the drum after one pass through the electrolyte. Means for washing the plated surface of the drum as it emerges from the electrolyte, before stripping, is provided. The heat of the drum assures complete drying before winding.

About 175 lb. of copper is deposited on each drum daily. The sheet produced measures approximately 0.00135 in. thick. One roll is removed daily. The approximate length of a 31-in. sheet is 1120 ft. The surface

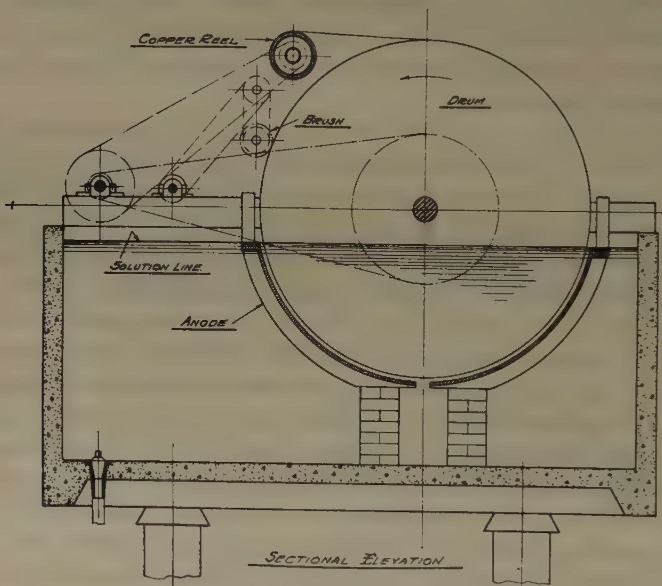

FIG. 1.—SECTIONAL ELEVATION OF FIRST-STAGE MACHINE.

of the sheet next to the drum has a bright satin finish, while the other side has the characteristic appearance of deposited copper. The finished roll is ready for the next operation, or it may be trimmed and shipped without further treatment.

In the second stage a roll of the 1-oz. material produced on the drum is built up to any desired thickness. The sheet is conducted through a series of depending loops (Fig. 2) suspended between adjacent anodes spaced along the length of the tank. These anodes are the standard crude anodes used in the regular refining operation. The flow of current through the tank is constant and the thickness of the sheet is determined by its rate of movement. About 350 lb. of copper is deposited in each tank daily.

The rollers at the top of the tank serve as the cathode contacts. Each roller has a flange that revolves in a mercury trough cut in the busbar mounted on the frame. A shallow pan is mounted under each

roll, and water flows through them continuously. This serves to maintain a clean contact between the moving sheet and the contact rollers.

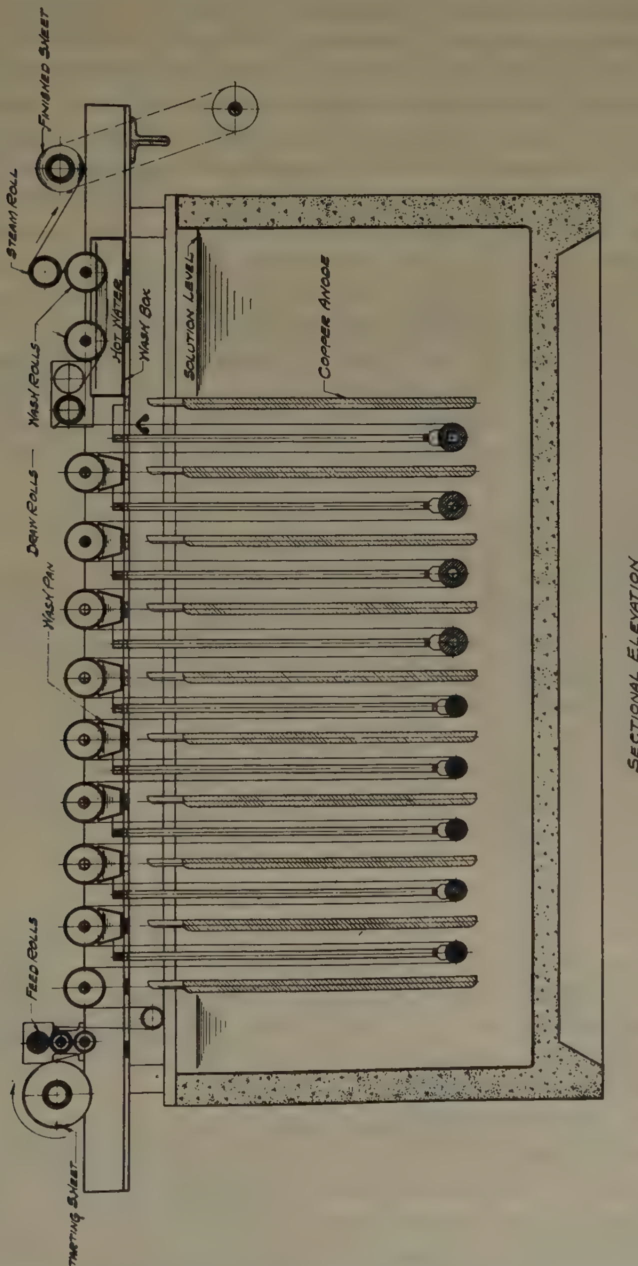

FIG. 2.—SECTIONAL ELEVATION OF SECOND-STAGE MACHINE.

The sheet is moved by draw rollers at the end of the tank. Means for washing and drying before winding are provided. The sheet from the

drum is soldered to the end of the sheet in the loop tank. This takes only a few minutes and may be done without stopping the machine. The finished sheets need only to be trimmed or otherwise cut for sale.

The process is peculiarly adapted to operation in a copper refinery. The use of lead anodes in the drum section reduces the number of liberator tanks necessary in the regular refinery operation, whereas crude copper is refined in the loop or thickening tanks. The tanks in each section are connected to the regular electrolyte system of the refinery. The electrolyte flows directly and continuously from the tank house through the drum section and back. Electrolyte for the loop section is withdrawn from and returned to the tank house intermittently. A reservoir with separate circulating system is provided for this section. Needless to say, careful control of acid and copper content of electrolyte, as well as temperature and rate of circulation, is necessary in both sections.

Change of anodes in the drum section is not necessary more than once a year. When it is necessary to change anodes in the loop section, the whole frame is lifted from the tank by an overhead crane. The tank is then drained and the slime pumped to the silver refinery before fresh anodes are placed and the frame returned. This is a rapid operation involving comparatively slight loss in production.

The existing pilot plant consists of six drum tanks and six loop tanks. The drums are approximately 66 in. in diameter. Five of them are 31 in. wide and one 51 in. wide. There are two electrical circuits in the drum section, each circuit containing three drums in series. The normal current flow is 3000 amp., although the section may be and has been operated at 4000 amp. The current density ranges from 140 to 190 amp. per square foot. Five of the loop tanks are designed to thicken 31-in. material, while the sixth handles 41-in. material. These tanks are all in series and the normal current is 6000 amp., with a current density of 40 amp. per square foot.

The daily capacity of the plant is 3150 lb. deposited in 24 hr. This is divided into 1050 lb. of drum copper and 2100 lb. of loop copper. Of course, the pounds deposited per day does not represent actual production of finished material, as there is always a certain amount of scrap. At present scrap ranges from 15 to 20 per cent, although it has been much higher. Eventually, however, total scrap, including trimmings and cutting waste, should be well under 10 per cent.

The voltage in the drum section ranges from 2.0 to 2.8 while in the loop section it ranges from 0.5 to 0.8. The ampere efficiency is approximately 95 per cent. In the drum section approximately 0.8 lb. of copper is deposited per kilowatt-hour of direct current while approximately 1.8 lb. per d. c. kilowatt-hour is deposited in the loop section.

The sheet material produced is sound and uniform, both in physical properties and gage. The control of average thickness is positive,

depending simply upon speed and current flow, while localized variation is slight and well within the limits of commercial variation. The remarkable smoothness and uniformity of the product is due primarily to the fact that plating is confined to one side of the sheet and that it never passes beneath an anode. Physical tests indicate a slight grain effect lengthwise of the sheet; that is, tensile strength and elongation are greater lengthwise of the sheet than crosswise. The tensile strength ranges from 30,000 to 40,000 lb. per sq. in. and the elongation from 15 to 25 per cent, depending upon whether the test sample is cut crosswise or lengthwise of the sheet.

Problems in Development

The problems encountered in the development of the process have been numerous as well as difficult. First and most important is the surface of the drums. It is probable that lack of a suitable starting surface has been the most serious single obstacle in the earlier development of a process for the electrolytic production of thin metal sheets. Heavier sheets have been produced in this manner for various purposes in different places for some time, but this production has been limited, as there is no advantage in producing heavy sheets in this way except under peculiar circumstances. While the surface of the drum is relatively unimportant with this class of work, it is most important when an attempt is made to produce wide sheets in the foil class for any length of time. Almost any suitable metal surface properly polished may be used successfully for a few hours or even a few days, but continuous operation month after month, and even for several years, is quite a different matter. The surface is subjected to extreme corrosive conditions as it enters the bath and the mechanical effect of stripping is considerable. After trying many different metal surfaces, including copper, silver, chromium, stainless steels, and even tantalum, to mention only a few, lead was finally decided upon as the most promising. Lead alone, however, may not be used successfully for more than a few days. This time may be prolonged by the use of suitable well-known stripping agents, but it was realized eventually that the solution of the problem of uninterrupted continuous production for a long period did not lie in this direction. Finally a mechanical means of continuously treating the surface of the drum was developed. This is a grinding and polishing operation, carefully controlled. In this manner a new metallic surface is presented to the action of the electrolyte for each revolution of the drum. The lead surface of the drum is one inch thick and may be used continuously for about two years before renewal is necessary.

Of almost equal importance is the treatment of the edges of the drum. Unless means to prevent are provided, the edges of a sheet produced in this manner are thick and ragged. This again is negligible when heavy

metal is produced, but suitable treatment of the edges of the drum is vital when it comes to the production of very thin sheets. The use of stop-off media, such as varnish or asphaltum, to define the plated area, as well as the use of "thieves" attached to the cathode to control thickness, is old in the art of electroplating. Similar means have been tried in the electrolytic production of sheet metal in the past. All these were tried during the development of the present process, with more or less success for a comparatively short time, but none was found to be suitable for long-continued operation. Finally a combination of mechanical and electrical means for accomplishing the desired result was developed. Increased thickness at the edges of the sheet is prevented by means of shields that obstruct the flow of current from the edges and back of the anodes. A clean-cut edge, which may be stripped without tearing, is produced by secondary cathodes suspended on the tank walls, so connected that they are cathodic to the ends of the drum. The solution at the ends of the drums is separated from the main body of electrolyte by means of the shields referred to. Therefore, the surface of the ends of the drum is always anodic to the electrolyte in contact with it, and no plating on it is possible, while the cylindrical surface of the drum, upon which the sheet is deposited, is cathodic to the main body of electrolyte. In this manner plating over the edges of the drum is prevented and clean-cut edges are produced on the sheet.

The greatest trouble in the development of the second or thickening stage of the process was the prevention of wrinkles. This is a matter of guiding and tension control. Finally a very simple means of automatically controlling tension was developed. The cathode contacts consist of copper rollers at the top of the frame, mounted loosely on the shafts that support them. Each shaft is provided with a sprocket secured by means of a set screw. In actual operation certain sprockets are set for driving while others idle, the number of each varying according to the particular weight of material being produced. Where the shafts revolve there is no friction between them and the contact rollers. On the other hand, where a tendency toward excessive tension exists, the revolving shafts exert a driving effect upon the rollers and excessive tension is avoided. Friction is introduced where the sprockets are not set and the shafts do not revolve. This is necessary in order to avoid slack in the sheet. The actual manipulation of the sprockets is the result of long experience, needless to say.

Another device to prevent wrinkles is necessary when copper weighing 3 oz. or less to the square foot is produced. In this connection it should be remembered that the contact rollers revolve in pans of running water, necessary to keep them clean. This produces a capillary effect between the surface of the sheet and that of the rollers, which has a decided tendency to produce wrinkles in the sheet. This tendency is avoided by

reducing the area of contact between the sheet and the rollers, by means of small auxiliary rollers above and to one side of the center line of the larger contact rollers. Strangely enough it has been found that the use of soft rubber rollers at the feed end of the tank is necessary, while hard rubber rollers are used at the other end.

There were many other difficulties of almost equal importance, which were overcome.

Uses for Sheet Copper

The intensive development of the process itself was preceded by considerable experimenting with thin copper in combination with other materials, particularly asphalt and asphaltic materials. A large part of this work was directed toward the building field. One of the most promising uses appeared to be the substitution of thin copper for felt in built-up roof construction. In various combinations with other materials, such as asphalt-saturated felt and fabric, insulating board, etc., its use for other forms of roofing, including shingles, as well as flashing, sheathing and general damp-proofing, appeared feasible. The results of these tests, most of them having run for several years, are most gratifying. The built-up roof constructions have proved very satisfactory and several large roofs have been applied during the past year. It is evident, on the basis of these applications, that a roof of this type may be applied in competition with the better grades of built-up roofs.

Another interesting possibility developed during this period is an insulating shingle. Shingles consisting of insulating board protected by thin copper on the exposed area and edges and asphalt-saturated felt on the unexposed areas, were exposed to the weather at several widely separated places throughout the country over five years ago. As a result of these tests the roofs of two houses were covered with these shingles three years ago. The oldest decks were examined recently and showed negligible effect of the weather. The shingles present a handsome, massive effect when applied, although actually they are light in weight. Application is rapid and economical. When allowance is made for their insulating value, such shingles would provide a comparatively cheap roof. Moreover, recent official tests indicate that they provide substantial fire protection.

To meet the demand for color in residential roofing, several special paints were developed concurrently with the development of the shingle itself, and have been on exposure from three to five years. No appreciable reduction in color value is noticeable. The colors include a rich red, two greens, one a remarkable approximation of natural patina, a copper brown and a slate gray. Deposited copper provides an excellent base for paint because it is absolutely free from grease.

It is evident from experience that the original expectations as to its possible use in new fields in substantial volume will probably be realized. During the past two years it has been tested by many people throughout the country for almost as many purposes, both alone and in combination with other materials, among these purposes being:

In combination with asphalt-saturated roofing felt, as a cap sheet for built-up roofs and, with heavier copper, as a flashing for asphalt roofing, the copper surface protecting underlying asphalt and felt from sunlight and moisture and preventing the formation of blisters between the felts; and also as a protection against fire.

In the more flexible forms—that is, in combination with fabric or reinforced paper—as spandrel flashing for damp-proofing inside walls. For roofing railway cars, ferry boats, etc., and for making automobile tops and fireproof awnings.

In combination with stiffer materials, such as steel (to which it may be attached either by soldering or with asphalt), plywood or insulating board, for use in building construction and refrigeration.

In combination with paper, for bookbinding, fancy boxes, etc. The satin finish of the drum copper is an excellent base for printing or lithographing.

Probably the greatest single use for the copper alone is in the roofing and general damp-proofing fields, as already mentioned. Experience has demonstrated that the bond between asphalt and copper is maintained, and contrary to early fears, expansion and contraction difficulties have not developed.

This new form of copper is distributed by The American Brass Co. under the name of Anaconda Electro-Sheet. In 30 and 40-in. widths it is made in a range of weights running in even ounces from 1 to 8 oz. to the square foot. The lighter material, weighing 1 or 1½ oz. to the square foot, may be obtained in 50-in. widths. Lengths up to 1800 ft., depending upon thickness.

A Comparison of the Use of Various Fuels in Copper-refining Furnaces

By E. S. Bardwell,* Great Falls, Mont.

(New York Meeting, February, 1932)

The reverberatory copper-refining furnaces at the Great Falls Reduction Dept. of the Anaconda Copper Mining Co. have used successively as fuel, lump coal on grates, pulverized coal, oil and natural gas. In no other plant, so far as can be ascertained, has such a variety of fuels been used. This fact affords a unique opportunity for comparing the relative efficiency obtained with these several fuels under nearly identical conditions.

Furnaces

The present furnace refinery at Great Falls was placed in operation in April, 1916. The original installation consisted of two furnaces, each with hearths 36 by 14 ft. and provided with fireboxes 10 by 12 ft. for firing lump coal on grates. These furnaces had air-cooled sand bottoms and chrome-brick side walls both above and below the metal line. The roofs were of 12-in. silica brick. Each furnace was provided with a waste-heat boiler of approximately 400 boiler-hp. capacity.

We soon found that chrome brick was very unsatisfactory as side-wall material because of its excessive spalling. At the first opportunity the furnaces were rebuilt, using magnesite brick below the metal line and silica brick from the metal line to the roof. At the same time the thickness of the side walls was increased, thus decreasing the width of the hearth to 13 ft. At about this time also, the thickness of the furnace roofs was increased from 12 to 15 in. No further changes occurred in furnace design until 1922, when the use of lump coal on grates was abandoned in favor of pulverized coal.

The first charge to be melted and refined with pulverized coal was cast on March 30, 1922. When pulverized coal first was used the burner pipes were introduced directly into the firebox as it was felt that some kind of a combustion chamber might prove necessary, but this was not needed. The fireboxes were dismantled and the burners introduced through openings directly over the bridge wall. During the summer of 1922 one of the furnaces was completely rebuilt, lengthened 10 ft. and provided with a silica-brick bottom in the form of an inverted arch. At

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the same time the thickness of the furnace roof was increased from 15 to 20 in. The other furnace was not remodeled until nearly two years later, at which time it also was lengthened and the width increased by 18 in., giving a hearth 45 ft. 7 in. by 14 ft. 6 in.

In 1926 a third furnace was constructed having the same general dimensions as those last mentioned. Recently the furnace first lengthened has been reconstructed and widened. By a slight modification in design the length of the hearth of this furnace was increased to 46 ft. 10½ in. The hearth area of this furnace has been successively increased from 433 to 528 sq. ft. and finally to 600 sq. ft. Thus, we now have two

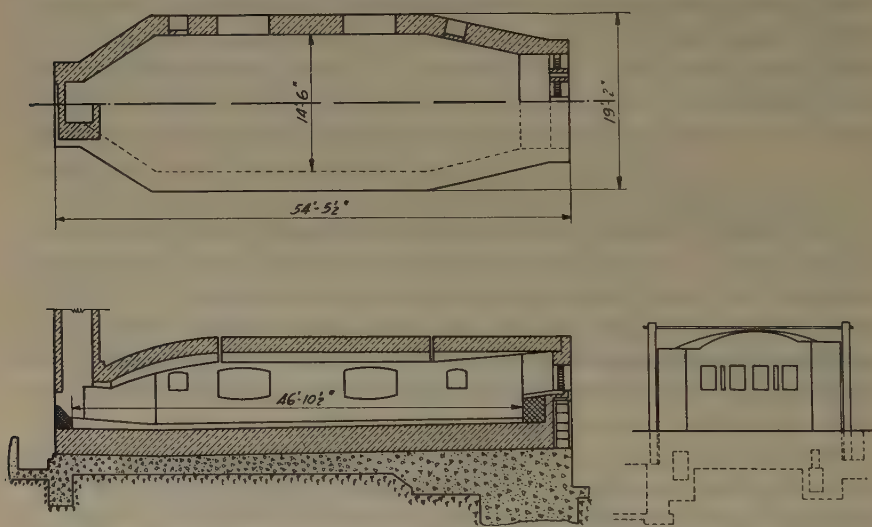

FIG. 1.—COPPER-REFINING FURNACE; HEARTH AREA, 600 SQUARE FEET.

furnaces with about 580 sq. ft. of hearth area and a third with approximately 600 sq. ft. of hearth area. Fig. 1 shows the present dimensions of the largest of the three furnaces.

FIRING WITH COAL ON GRATES

Lump coal for grate firing was delivered by a Link-Belt conveyor into bins directly over the furnace fireboxes, whence it was charged through openings in the roof provided for that purpose.

At first a local coal known as Lochray, high in ash and varying from 3 to 5 per cent in sulfur content, was used. So many difficulties were experienced in using a fuel of this character that we soon resorted to lower sulfur coals from Wyoming, and Bear Creek coal from the Red Lodge field of Montana, although mixtures of local coals with Wyoming and Bear Creek coal were used with a fair degree of success from time to time.

Table 1 gives typical analyses of the Lochray, Wyoming and Bear Creek coals used at the furnace refinery during this period.

The average production per furnace day during the entire period of using lump coal on grates was 338,400 lb. If, moreover, we leave out of account the operations of the first few months and a period during which the furnaces were being alternated with no attempt at making a 24-hr. cycle, 340,000 lb. per furnace day would represent very closely the average daily capacity of the furnaces as they were then operated.

TABLE 1.—*Typical Analyses of Coals Used for Firing*

	Lochray Lump Coal	Wyoming Lump, Diamondville Field	Bear Creek Coal, Montana
Moisture, per cent	5.5	3.3	6.3
Combined H_2O, per cent	1.9	0.3	2.8
Volatile combustible, per cent	25.6	32.8	33.2
Fixed carbon, per cent	49.3	58.9	46.7
Ash, per cent	18.7	4.7	11.0
Sulfur, per cent	3.9	1.0	1.0
British thermal units, dry	10,620	13,500	11,990
British thermal units, wet	10,030	13,122	11,250

The month of August, 1916, gives the best index of what it was possible to accomplish with Lochray coal on grates. During this month the furnace was operated continuously using Lochray coal. The furnace produced 292,800 lb. per furnace day and the average elapsed time per cycle was 23 hr. 50 min. distributed between the several furnace operations as follows:

	Hours	Minutes
Charging	2	0
Melting	10	30
Rabbling	5	0
Poling	3	0
Casting	3	20
Total	23	50

During this period we obtained a fuel ratio of 3.12 tons of copper per ton of coal, which, reduced to British thermal units per pound of copper, is equivalent to 3210.

Two difficulties were encountered in the use of this fuel. First, the high sulfur and high volatile content of the fuel made necessary a strongly oxidizing atmosphere in the smelting chamber at all times.

This was, of course, particularly true whenever it became necessary, as was often the case, to replenish the fire during the latter part of the poling period in order to carry the heat through the time of casting. In this case particular care was required to avoid filling the furnace with reducing gases. Numerous charges were overpoled by reason of failure or inability to satisfactorily attain such an end. When it became necessary to charge fresh coal at such times the blower air was shut off and auxiliary air supplied directly over the bridge wall. This served in most cases to keep the furnace atmosphere clear while the volatile matter from the fresh charge of coal was being given off. This having been accomplished, the blower air was again turned on and furnace operations resumed.

The second difficulty arose from the fact that each day after the charge was out it became necessary to pull the grate bars and remove all clinker from the firebox. At the end of each furnace cycle the firebox contained nearly 12 tons of ash and clinker. The ash and clinker resulting from the use of Lochray coal contained nearly 25 per cent of fixed carbon, which in itself was a source of considerable loss. Samples of ashes and clinker taken at this time showed as high as 3400 B.t.u. per pound; in other words, between 8 and 9 per cent of the total calorific value of the fuel used was wasted in ash and clinker removed from the firebox. During the time that the firebox was being cleaned and made ready for the next cycle of operations the furnace was being charged. Under these conditions the furnace became quite cold, which naturally tended to promote spalling of refractories and resulted in high repair costs.

The use of Diamondville and Bear Creek coals, lower in both sulfur and ash, brought about a decided improvement. The fuel ratio was increased and larger charges were obtained from the furnaces but the highest fuel ratio maintained over any considerable period was 4 tons of copper per ton of coal. Regardless of the fuel used or the average size of the charges handled, the average heat required per pound of copper remained practically constant throughout the entire period at 3225 B.t.u.

So great were the difficulties attendant upon the use of coal on grates that during the latter part of March, 1922, equipment for firing with pulverized coal was installed.

Firing with Pulverized Coal

The use of pulverized coal for firing the reverberatory copper-refining furnaces at Great Falls was discussed[1] in a paper presented at the Salt Lake meeting of the Institute in September, 1925, and need only be summarized here.

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[1] E. S. Bardwell and R. H. Miller: Pulverized Coal as Fuel for Copper-refining Furnaces. *Trans.* A. I. M. E. (1926) **74**, 592.

As equipped for pulverized coal each furnace was provided with three burners of the Warford type. Coal pulverized so that 96 per cent passed the 100-mesh and 80 per cent passed through the 200-mesh screen was transported in specially designed cars from the pulverizing plant and forced by means of air under pressure from the cars into bins located above the furnaces at the burner end. Typical analyses of the fuels used in pulverized form are given in Table 2. Both of the fuels were entirely satisfactory from every point of view.

TABLE 2.—*Typical Analyses of Pulverized Fuels*

	Moisture, Per Cent	Volatile, Per Cent	Fixed Carbon, Per Cent	Ash, Per Cent	Sulfur, Per Cent	British Thermal Units	
						Dry	Wet
Utah (Sunnyside slack)..	5.0	33.6	53.0	8.4	0.8	12,757	12,190
Wyoming (Diamondville slack)................	5.4	36.0	50.9	7.7	1.1	12,689	12,010

The use of pulverized coals brought about a marked increase in both the production per furnace day and the fuel ratio. The average pounds per furnace day was increased from 338,400 to 440,000 lb. This increase, however, is due in part to the lengthening of one of the furnaces from 36 to 45 ft. 7 in., with resultant increase in hearth area from 433 to 528 sq. ft. The fuel ratio increased with the use of pulverized coal from a maximum of 4 tons of copper per ton of coal to 6.8 tons of copper per ton of coal.

One point which was not mentioned in the paper referred to above is the effect brought about on fuel ratio by reason of the lengthening of one of the furnaces. Lest it be thought that the rebuilding and lengthening of this furnace, which took place only shortly after we began to use pulverized coal, accounts entirely for the increase in fuel ratio it may be well to state that prior to lengthening the furnace we used 309 lb. of coal per ton of copper, whereas after the furnace was lengthened the average was 272 lb. of coal per ton of copper. The increased hearth area, however, naturally brought about an increased production per furnace day. A comparison of the two furnaces as operated during the first six months of 1923 shows the average daily capacity of the shorter furnace to be 391,536 lb. as against 470,648 lb. per day for the longer furnace. The average coal consumed per ton of copper produced during the first six months of 1923 was 294 lb., or 1867 B.t.u. per pound of copper; a reduction of nearly 42 per cent in B.t.u. per pound of copper over grate firing.

While there was an increase of approximately 28 per cent in slag production, it was almost entirely due to ash settling on the charge, as the

percentage copper contained in the slag to copper in the charge increased only 3.4 per cent, or from 2.04 tons of copper in slag per 100 tons of copper charged to 2.11 tons.

The period of operation with pulverized coal was rather too short to show just what the effect of using pulverized fuel might be on furnace repairs. It may be stated, however, that repairs were no more than with coal on grates, and in all probability they would have averaged at least 10 per cent less had we continued to operate with pulverized coal.

The change from pulverized coal to oil firing was due primarily to the opening of the Kevin-Sunburst oil field in the north central part of Montana, which made it possible to obtain fuel oil at a reasonable cost.

Use of Fuel Oil

The first charge using fuel oil was cast on Aug. 18, 1923, and from that time until May, 1928, oil was used continuously.

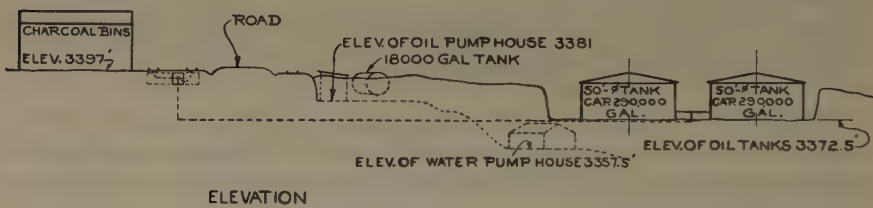

Fig. 2.—Oil storage.

Fig. 2 shows the general arrangement of storage facilities and pump houses at the plant. The oil-unloading station is indicated in this drawing between the coal and charcoal bins and the road and is served by two standard-gage tracks. From this station the oil received in tank cars could be unloaded either to the two 50-ft. storage tanks or to the 18,000-gal. supply tank located immediately in front of the oil-pump house. From a point just north of the oil pump house a tunnel (not shown) extends under the road and beneath the coal and charcoal bins into the furnace refinery building. It was through this tunnel, originally a part of the conveyor system used during the days of coal firing, that our oil lines from the pump house entered the furnace refinery building. A Worthington 7½ by 5 by 6-in. steam pump in the water pump house served to pump oil from the storage tanks to the supply tank or to other departments in which oil was used, or, through the means of this pump oil could be reloaded into tank cars for transfer to boiler plants not directly connected through pipe lines.

The two 50-ft. storage tanks were provided with steam coils and during the winter season enough steam was, and in fact still is, kept on these tanks to keep the oil warm enough so that it will flow. This is done because, while the entire plant is equipped at present for natural gas,

oil is our stand-by in case interruptions should occur in service. The oil in the supply tank was maintained at 130° F. at all times and was delivered to the oil pump house at this temperature.

In the oil pump house there were three Gould, vertical, single-acting Triplex plunger pumps, 3-in. dia. by 4-in. stroke, each direct connected to a 5-hp. General Electric motor (600 r.p.m., 440-volt, three-phase). Each of these pumps has a capacity of 25 gal. per minute at 150 lb. pressure. A steam pump of suitable capacity was provided for emergency use.

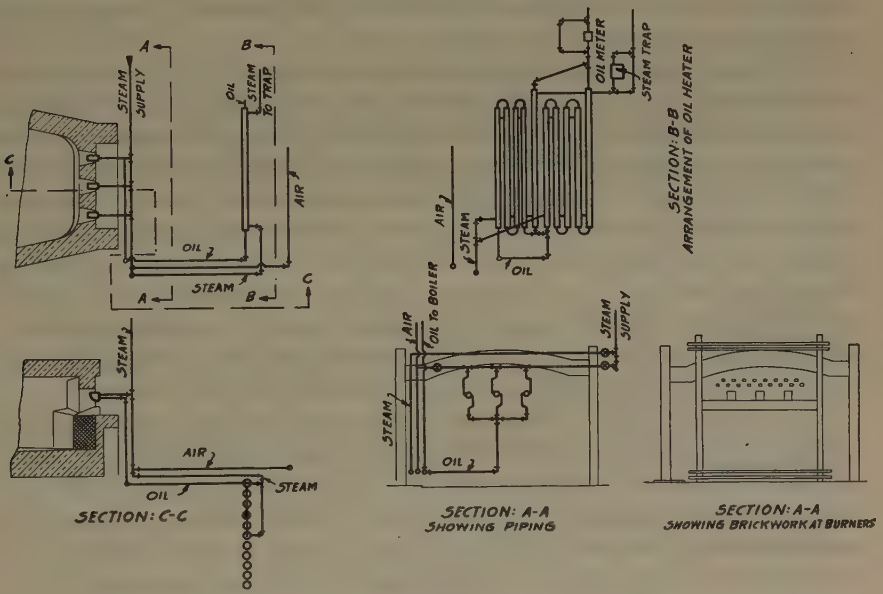

Fig. 3.—Arrangement of burners.

The 1¼-in. oil line to each furnace was equipped with a "Sarco" oil strainer and a ⅝-in. Worthington Duplex piston oil meter. Between the meter and burners the oil passed through a steam-coil heater which served to raise the temperature to 180° to 200° F. for delivery to the burners.

Each furnace originally was provided with three No. 2 Best burners mounted on swivel joints and so piped that either air or steam could be used for atomizing. In practice, air was always used except in cases of emergency.

The general arrangement of burners is indicated in Fig. 3, which also shows a sketch of the oil heater. The drawing also shows the checker directly over the burner ports, consisting of 15 openings in the burner wall. These openings were kept wide open during the melting period and served to protect the roof of the furnace at this point. During the poling period they were closed with brick and luted up. In order to

restrict the amount of secondary air admitted through the burner ports during the poling period, slotted iron plates were slipped down over the burner pipes so as partly to close the openings around the burners. This had the effect of giving a slightly reducing atmosphere in the smelting chamber at such times.

After having operated in this manner for several years, the two outside burners were moved toward the center about 10 in. and the use of a center burner was discontinued. With two burners instead of three to be watched and adjusted by the furnacemen, more uniform operating conditions were attained, which resulted in some increase in fuel economy and reduced wear on furnace walls.

The pressure of the oil at the burners was 50 lb. and 80-lb. air was used for atomizing the oil. It was extremely important that the air pressure should be maintained as constant as possible and special arrangements were made to assure that this should be the case. Unfortunately, the air was not being metered at that time, but we know that a compressor with a capacity of 1000 ft. of free air per minute would just supply six burners at the maximum rate at which oil was being burned during the melting period. The oil pressure was maintained at 50 lb. by means of a pressure relief valve located in the pump house. Too much stress can hardly be laid on the necessity of maintaining constant pressure of both oil and air at the burners if maximum efficiency is to be obtained.

The question may be asked as to why air rather than steam was used for atomizing the oil. Andros[2] states that numerous tests have proved that with steam at 80-lb. pressure and air at 80-lb. pressure there is a saving of 12 per cent in fuel if air is used instead of steam. Inasmuch as waste-heat steam was available, it may be thought that it would have been cheaper to have utilized steam instead of air for atomizing. The waste-heat steam, however, was available for heating electrolyte in the tank room at the electrolytic copper refinery, where it replaced coal-fired steam costing at least $12 per boiler-horsepower month to generate. The steam pressure, moreover, was too low for satisfactory use in atomizing oil; being at the time in the neighborhood of 45 lb. We estimate that an average of 25 boiler-hp. would have been required for each furnace had steam been used instead of air. On the other hand, the total cost of operating a 1000-ft. compressor capable of supplying not only air for atomizing oil but air for rabbling, painting molds and operating hoists averaged only about $340 per month during this period. In view of this the use of air for atomizing not only saved fuel but was more economical under existing conditions than the use of steam would have been.

During the greater part of the time that oil was used as fuel on our furnaces our source of supply was the Kevin-Sunburst field. Following

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[2] Andros: Fuel Oil in Industry, 155.

is an analysis of this oil: sp. gr., 0.928; carbon, 85.1 per cent; hydrogen, 12.9; sulfur, 1.9; nitrogen, 0.1; B.t.u. per lb., 19,224 (calculated).

It will be remembered that when we began to use oil we had two furnaces, one with a hearth area of 433 sq. ft. and the other with a hearth area of 528 sq. ft. It would be of interest to compare the performance of these two furnaces on oil in the same manner as they were compared when using pulverized coal. Such a comparison, however, is misleading because the smaller furnace had become badly encrusted, so that the capacity had been much reduced. The larger furnace, however, was producing at this time 450,000 lb. per day with an oil consumption amounting to 20 gal. per ton of copper.

In September, 1924, the smaller furnace was rebuilt, the hearth area having been increased from 433 to 580 sq. ft. Trouble was experienced with both furnaces because of the tendency towards formation of incrustations on the bottoms, which soon reduced the capacity of the furnaces. It was determined ultimately that too much air was being circulated through the furnace bottoms. The year 1925 was devoted to working out some difficulties and is not truly representative of what could be accomplished.

Table 3 shows the fuel efficiency obtained on our furnaces for the years 1925, 1926 and 1927 with oil fuel. Comparing the furnaces, it

TABLE 3.—*Fuel Efficiency with Oil*

Year	No. 1 Furnace (580 Sq. Ft. Hearth Area)				No. 2 Furnace (528 Sq. Ft. Hearth Area)			
	Pounds per Furnace Day	Number of Charges	Oil per Ton Copper, Gal.	B.t.u. per Pound Copper	Pounds per Furnace Day	Number of Charges	Oil per Ton Copper, Gal.	B.t.u. per Pound Copper
1925	497,256	246	19.9	1,479	455,354	169	22.4	1,664
1926	498,939	245	19.4	1,441	472,300	245	20.4	1,516
1927	485,427	184	19.7	1,464	476,112	244	19.9	1,479
Avg. total	494,643	675	19.7	1,464	469,362	658	20.7	1,538

will be seen that No. 1 furnace has approximately 10 per cent greater hearth area than No. 2. The average daily production of No. 1 furnace was 5.4 per cent greater than No. 2 while the fuel efficiency indicated as British thermal units per pound of production is about 5 per cent better in the case of the larger furnace. The fuel consumption here, as elsewhere in this paper, unless otherwise stated, is gross fuel consumed, and includes all fuel used for heating furnaces as well as for operating.

Table 4 gives the elapsed time for the several operations into which the furnace cycle may be divided, together with the total average gallons of oil consumed during each period and the corresponding rate of fuel consumption.

Table 4.—*Data on Operations of Furnace Cycle*

Operation	Elapsed Time		Fuel Oil Consumption	
	Hours	Minutes	Total Gallons	Gallons per Hour
Charging	1	57	249	128
Melting	10	0	2,952	295
Rabbling	5	10	901	174
Poling	2	47	93	33
Casting	4	6	304	74
Total	24	0	4,499	187

In order to get some idea as to how pulverized coal and oil compare from the point of view of fuel efficiency, we may compare the results obtainable on the longer furnace only with the same furnace operating with oil as fuel. On this basis we find that the average daily capacity using pulverized coal as fuel was 470,648 lb. as against 469,362 lb. with oil. The heat units per pound of production were 1621 B.t.u. with pulverized coal as fuel as against 1475 B.t.u with fuel oil. In other words, when the change was made from pulverized coal to oil it resulted in about 10 per cent of saving in heat required per pound of marketable shapes produced. A considerable part of this saving is due to the fact that when using pulverized coal there was a blanket of ash on the bath of metal in the furnace, which resulted in less efficient heat transfer.

We attribute the excellent results obtained with fuel oil to:

1. The use of air for atomizing.

2. The installation of instruments to record oil temperature and pressure at the burners; air pressure, stack temperature, etc., and the training of operators to watch these and all other essential factors affecting furnace operation.

3. Steady operation seven days a week on a regular 24-hr. cycle.

Natural Gas as Fuel

In April, 1928, a pipe line about 100 miles long was completed, connecting the natural gas field near Shelby, Mont., with Great Falls. The wells in the Shelby field deliver gas to the main regulating station in the field at rock pressures varying from 750 lb. per sq. in. down to 240 lb. per sq. in. The volume is controlled at the wells by means of control disks but no automatic pressure control is provided. The average field pressure varies from 275 lb. per sq. in. in winter down to 200 lb. per sq. in. in summer. From the main regulating station in the field to Telstad, a distance of 10 miles, the line is 10-in. From Telstad to Great Falls a 12-in. line is provided. At the Great Falls end of the line pressures vary

from 100 lb. per sq. in. during the summer to 200 lb. per sq. in. during the winter. Thus far no booster stations have been found necessary and none have been provided.

A branch from the main gas line serves the Great Falls Reduction Dept. of the Anaconda Copper Mining Co. Where the line enters the company's property, settlement meters are located and the pressure is reduced to from 25 to 30 lb. per sq. in. for distribution to the various departments where gas is used. Screens for removing debris of various kinds from the line are also provided.

Gas is received at the furnace refinery through a 6-in. line at 24 to 30 lb. per sq. in. pressure. There is a meter house just outside the refinery building where the gas is metered by means of a Westcott orifice meter and the pressure further reduced to 5 lb. per sq. in., at which pressure it enters the furnace refinery building through a 12-in. welded pipe. This 12-in. pipe runs lengthwise of the furnace building, suspended from the roof trusses. It is directly over the charge track.

Fig. 4.—Mettler entrained combustion gas burner.

The burners at each of the three furnaces receive their gas through 8-in. lines dropped from the main 12-in. line suspended from the roof trusses. Each furnace is equipped with a Westcott orifice meter. The charts from these meters are planimetered daily and the totals compared with the master meter. The totals of the various meters are similarly compared daily with the settlement meter. The gas volume shown by the meters is reduced to 60° F. and 8 oz. above 14.4 lb. per sq. in., which is the settlement basis.

Inasmuch as natural gas had never been used in reverberatory copper-refining furnaces prior to the installation at Great Falls, we had little to guide us in the application of this fuel to our furnaces. For several months homemade burners of various designs were used and while no great difficulty was experienced the gas consumption was higher than we had expected and we found it necessary to reduce the size of our charges somewhat in order to make the regular 24-hr. cycle.

After considerable experimenting with burners, we tried and adopted the Mettler entrained combustion gas burner, which has proved entirely satisfactory for our purpose. At first four 38-hole burners were used on each furnace. Fig. 4 shows one of these burners with the adjustable shutter, used for controlling the flow of air through the burner openings, moved back so that the gas manifold can be seen. Each of the 38 openings in the gas manifold is provided with four jets set 90° apart

and directed inward at an angle, so as to give an injector effect calculated to pull air in through the burner openings. The face of the gas manifold is provided with a refractory block having openings corresponding to those in the gas manifold. The streams of gas from the four jets in each mixing tube meet at a central point and, impinging one against the other, cause violent agitation of the gas in the solid core of entrained air. The length of the burner tubes is such that the firing of the mixture takes place before the gas and air have a chance to separate again. This results in complete and almost instantaneous combustion with a minimum of excess air.

The manufacturers of these burners equipped them with high-alumina refractory blocks which were quite unsatisfactory for use in a copper-refining furnace. At present we are using a rammed block consisting of silica sand and chrome cement, half and half, with 20 per cent of sodium silicate used as a binder. The life of these rammed blocks is about 60 days.

The gas manifold is 17½ by 26 in. As originally installed, four of these burners were set on end equally spaced at the burner end of the furnace. At present three burners only are being used on each furnace. These burners are set on the side, as shown in Fig. 4, and are spaced 45 in. on centers with ports between to allow oil burners to be introduced should interruptions in gas supply occur. A 3-in. pipe supplies gas to each burner through a suitable regulating valve.

The natural gas used at Great Falls has the following analysis: Ethane 4.9 per cent; methane, 92.0; carbon dioxide, 0.4; undetermined, 2.7; total, 100.0.

Table 5 shows the total elapsed time for the various operations into which the furnace cycle is divided, together with the average rate of fuel consumption.

TABLE 5.—*Elapsed Time and Fuel Consumption*

Operation	Elapsed Time		Gas Consumed, Cu. Ft.	
	Hours	Minutes	Total	Per Minute
Charging	2	10	54,000	415
Melting	10	35	430,000	677
Rabbling	4	55	182,400	618
Poling	2	25	37,000	255
Casting	3	20	49,000	245
Totals	23	25	752,400	
Average				535

Each burner is provided with a manometer. The gas pressure at the burners is about 8 oz. during the melting period. During this period it is necessary to use a high-pressure air jet at each burner. About 300

cu. ft. of air per minute is required for each furnace (80-lb. pressure), which is introduced through ¼-in. pipes inserted into one of the openings in each burner.

The average production per furnace day since gas has been used has been 478,000 lb. and the average gas consumption, including gas required for heating furnaces, has been 3530 cu. ft. per ton of shapes produced. During the past year the average production per furnace day has been 498,000 lb. and the fuel consumption has averaged 3321 cu. ft. per ton of shapes produced. These figures for gas consumption include all gas used for heating furnaces as well as operating gas. The average cubic feet of gas used for operating alone per ton of shapes produced over a period of three years has amounted to 3247 cubic feet.

Comparison of Pulverized Coal, Oil and Natural Gas

In attempting to compare pulverized coal, oil and natural gas, the question arises as to the basis on which these fuels may properly be compared. The analysis of the several fuels has already been given. If we take as a basis first the metallurgical calorific value computed from the analyses on the assumption that the water vapor resulting from combustion remains uncondensed and cold, we have the following: pulverized coal, as purchased, 11,960 B.t.u. per pound; fuel oil, 19,224 B.t.u. per pound; natural gas (settlement basis), 983 B.t.u. per cubic foot.

On this basis 1 gal. of fuel oil would be equivalent to 12.42 lb. of coal, or 151 cu. ft. of gas. On this basis, if an average of 20.1 gal. of oil were required per ton of copper produced, we might assume that we would require 250 lb. of coal, or 3035 cu. ft. of natural gas.

This is not correct, however, because it may be shown by computation, taking into consideration the analysis and temperature of flue gases in each case, that 46.7 per cent of the heat units developed from the combustion of pulverized coal is available for furnace work while with oil 44.7 per cent is available and with natural gas only 41.9 per cent. Compared on this basis, 1 gal. of oil would be equivalent to 11.89 lb. of coal or 161 cu. ft. of gas. On this basis, assuming 20.1 gal. of oil per ton of copper produced, we might expect to use 239 lb. of coal, or 3240 cu. ft., of natural gas.

When pulverized coal was being used on our furnaces, as has been previously stated, the best average fuel ratio obtained was 6.8 tons of copper per ton of coal, or 294 lb. of coal per ton of copper produced. At this time we were operating two furnaces differing considerably in size. The coal consumption on the large furnace amounted to 272 lb. per ton of copper produced. Taking this figure for comparison, it will be noted that the fuel consumption which we obtained was about 15 per cent higher than might have been expected. The greater part of this increase we believe to have been due to the insulating effect of ash deposited on

the surface of the metal bath. When pulverized coal was being used, the amount of slag produced was 37 per cent greater than was the case when oil was used as fuel.

In each case the temperature of the gases entering the waste-heat boilers was approximately the same; averaging 1940° F., but varying from less than 1000° F. during charging to as high as 2500° F. during poling. The average temperature of gases entering the stack was 600° F. when pulverized coal and oil were being used as fuel. Since natural gas has been used, the stack temperatures have averaged about 800° F. This increase in stack temperature is due to increased volume and heat content of gases passing through our waste-heat boilers, which are not provided with baffles.

Table 6 gives comparative data concerning the use of pulverized coal, oil and gas. These figures are all based on the same furnace with hearth area of 528 square feet.

These figures represent a comparison of results obtained with pulverized coal, oil and gas on one of our furnaces. The fuel consumption represents that actually used for refining and does not include any fuel used for heating furnaces preliminary to a campaign. Figures previously quoted in this paper include fuel used for heating furnaces. It is difficult to understand why the coal required per ton of copper should be quite as high as it appears to be. The insulating effect of the ash from the coal, as has already been mentioned, accounts in part for the difference. The 272 lb. of coal per ton of copper is based on coal as purchased and includes shrinkage due to loss in moisture by reason of drying; also, dust losses in the coal pulverizing plant and any other losses that may have occurred. Taking 19.4 gal. of oil per ton of copper as standard and applying the ratio 11.89 lb. of coal per gallon of oil, it would seem that, even after allowing 15 per cent for the insulating effect of the ash, we should have required not over 266 lb. of coal per ton of copper. The remaining difference undoubtedly is due to inaccuracies in estimating the coal used and differences in operating conditions. We feel that were we to attempt again to use pulverized coal on our furnaces, we would obtain a better fuel ratio than is indicated above.

Taking 161 cu. ft. of gas as equivalent to 1 gal. of oil, we might expect to use 3123 cu. ft. of gas per ton of copper instead of 3397 cu. ft., which was the actual requirement in the case of the furnace used for comparison. One other refinery now using natural gas as fuel reports gas consumptions 10 per cent higher than expected. By some this has been attributed to the nonluminous flame obtained when gas is used as fuel. The author, however, believes that more likely it is due to differences in furnace operation or the placing of the burners. Regarding the latter point, it may be stated that the change recently made in the location of the burners has brought about a very definite improvement in the fuel ratio. The matter

Table 6.—*Comparative Data on Use of Pulverized Coal, Oil and Gas*

	Pulverized Coal	Oil	Natural Gas
Fuel analysis, per cent	H_2O 2.75 O_2 9.00 S 1.15 H_2 5.02 C 73.07 N_2 1.60 Ash 7.41 100.00	C 85.1 H_2 12.9 S 1.9 N_2 0.1 100.0	CH_4 92.0 C_2H_6 4.9 CO_2 0.4 Undet. 2.7 100.0
Metallurgical calorific value, B.t.u.	12,590 per lb. 11,960 (as purchased)	19,224 per lb.	983 cu. ft.
Average production, tons per charge	235	233	233
Average hourly rate of fuel consumption	2,663 lb.	188 gal.	32,983 cu. ft.
Fuel per ton of copper produced	272 lb.	19.4 gal.	3,397 cu. ft.
B.t.u. per pound of copper produced	1,627	1,441	1,670
Average temperature of gases leaving furnace	1,940° F.	1,940° F.	1,940° F.
Stack temperature	600° F.	600° F.	800° F.
Heat distribution percentage of B.t.u. in fuel:			
Available in furnace	46.7	44.7	41.9
Absorbed in boilers	38.9	40.2	36.6
Stack loss	14.4	15.1	21.5
Analysis of flue gases, per cent:			
CO_2	14.7	10.8	8.2
H_2O	6.6	9.9	16.0
SO_2	0.1	0.1	
O_2	2.2	2.1	2.2
N_2	76.6	77.1	73.6
Totals	100.0	100.0	100.0

of furnace operation has also been shown to be of importance. If, after it has become molten, the charge is allowed to lie dead in the furnace, more heat is likely to be required than would be the case were pipes introduced and the metal kept well mixed. In other words, it is easier to carry heat down into the metal bath than to drive it down. The figure given in the foregoing tabulation represents the average gas consumption actually obtained. The author is confident that the theoretical figure of 3240 cu. ft. of gas per ton of copper produced can be obtained in practice from the furnace of this size. On the larger furnaces having hearth area of 580 sq. ft. as against 528 sq. ft. in the case of the furnace chosen for comparison, even better fuel ratios have been obtained with charges of about 235 tons. During a recent month one of the larger furnaces produced 273 tons per furnace day with a gas consumption averaging 2780 cu. ft. per ton of copper produced.

While the decision as to what fuel is to be used in a copper refinery is likely to rest entirely on relative cost, natural gas may be said to be almost ideal from the operator's point of view. The cleanliness of gaseous fuel is a distinct advantage. It is easily regulated and once the burners are regulated little attention is required on the part of the operator.

DISCUSSION

(Francis R. Pyne presiding)

F. R. Pyne, Perth Amboy, N. J.—Mr. Bardwell's remarks about the decrease in the consumption of pulverized coal per ton of copper, due to the increase in the length of the furnace, are interesting. Doing a little calculating, we find that the increased production was obtained with a coal consumption of approximately 90 lb. per ton, which would indicate that a furnace having a long, shallow hearth is much more economical from the fuel standpoint than one with a short, deep hearth. That has been my own experience with furnaces.

I have noticed a tremendous difference in the slag produced with coal and with oil. With coal, it is a light, fluffy slag, more like a pumice, containing much less metallic copper than the slag made with fuel oil.

As Mr. Bardwell says, the decision as to what fuel to use is an economic one. It is interesting that about the time Great Falls was changing from coal to oil for economic reasons, another Anaconda plant, the Raritan Copper Works, was changing from oil to coal, with decided economy to both plants.

Mr. Boggs, what are they using in the Canadian refineries?

W. B. Boggs, Noranda, Que.—In Montreal, we are using fuel oil.

C. R. Hayward, Cambridge, Mass.—The author says that definite comparisons cannot be made because from time to time the situation of the furnaces and the shape of the furnaces were changed. He mentions the lengthening and widening of the furnaces, and also that the walls were made considerably thicker. He does not take the latter fact into consideration in comparing the efficiency of the fuel. I am inclined to think the thicker roof in particular may have improved conditions in the later work.

One thing is puzzling those who are using natural gas; that is, the luminosity of the flame. The author mentions it but indicates that he thinks it has little effect. At El Paso they were somewhat in doubt, before they began to use natural gas, as to whether they would be able to operate without illuminants in the gas. Much to their

surprise, they discovered that the furnaces apparently gave satisfactory results without enrichment of the gas with illuminants. We have been brought up to believe a luminous flame important in heat transfer. Is it really unimportant?

C. R. Pyne.—It would seem to me that a nonluminous flame would take longer to heat a furnace but that after it was heated, the equivalent of luminosity would be obtained from radiation from the hot brick.

A great many years ago, at Great Falls, the coal used was the Sand Coulee. The Lochray, used later, runs around 18 per cent ash; the Sand Coulee ran nearer 30 per cent and was very high in sulfur. The cathodes had to be dipped in lime before they were charged into the furnace to keep the sulfur down. They made a high grade of wire bar with low sulfur but a sloppy operation was involved in taking the lime-coated cathodes out and charging them in the furnace.

I have since seen the furnaces at Great Falls working with gas, and it is a nice, quiet, clean operation. From my experience with the other three types of fuel and my observation of the use of gas, I would say that, other conditions being equal as to cost, the gas would be much preferred. I would place pulverized coal next, because of its quietness; then oil, with its noisiness; nobody wants to go back to hand firing after once getting away from it.

A. Grönningsater, Sudbury, Ont.—As to the use of luminous gas, I have had a definite experience. Under certain conditions, poorer results are obtained without luminous gas. I will not go into the details of a long experiment, but will just mention that it had to do with an old-fashioned firebox from which originally was obtained a long, luminous flame; later changes were made, resulting in obtaining a shorter, nonluminous flame. We had analyses made and kept track of calories, and found that there was an increased fuel consumption. Fully utilizing a nonluminous flame depends upon the length of the furnace—whether it is long enough.

The remark was made that chrome brick was found unsatisfactory and that a change to magnesite was necessary. I would like to ask what the general experience has been in this respect.

F. R. Pyne.—Regarding the use of chrome brick, I can speak feelingly. In 1912, the manager of the plant of the U. S. Metals Refining Co. at Carteret wanted the furnace lined with chrome brick. It was lined with chrome and had a chrome roof, and finally all the furnaces were lined with it. It was all right until the time came to do something with the cobbings. Chrome was better than magnesite, but absorbed a terrific amount of metal. The cobbings could not be smelted. They formed a mush between the matte and the slag, which was very high in copper and kept filling up the space in the settler. There was either mighty little matte in the settler, or the mush overflowed and gave terrific copper loss in the slag. I think the copper went out in the slag in the long run. It was difficult to grind the stuff and treat it, so it was discontinued.

A. Grönningsater.—Your experience, then, was different from that at Great Falls, because there chrome brick did not stand up, evidently. That is what I was interested in. My experience seems to indicate that magnesite will stand up better than chrome brick but opinions seem to differ somewhat.

F. R. Pyne.—When we installed these chrome brick furnaces, the magnesite brick could not compare with the magnesite we get now. It was not particularly good and the lining of a furnace did not last at all. With chrome brick, we had no trouble as to lasting. The trouble was in treating the cobbings when we got through. At present, we are getting a remarkably fine result from linings with magnesite brick—far better than the results with the chrome brick.

A. GRÖNNINGSATER.—I wonder what the experience is nowadays with the present quality of chrome brick.

H. M. SHEPARD, Maurer, N. J.—It has been my experience that chrome spalls more than magnesite. We are using it at present in our verb arches and they last longer than the silica. When first fired on, they spall a great deal. At various times we tried them in the roof over the spots where we blow, with no success. What happened was that the silica was corroded off by the slag; the chrome spalled and we ended in the same predicament.

I think, Mr. Pyne, the author says something in regard to slag on page 462. I wonder if they have made any further improvement in that respect since using gas.

F. R. PYNE.—They have. The slag is now down to where it was when they were using oil.

E. S. BARDWELL (written discussion).—Mr. Hayward has brought out two points that require something in the way of explanation: (1) It is true that I outlined certain changes that took place in furnace dimensions, but this *was* taken into consideration in comparing the fuel efficiency. The figures given in Table 6 are based on a single furnace the dimensions of which, including thickness of furnace walls and roof, were the same. This fact, moreover, is specifically stated on page 462 of the paper. (2) As to the luminosity or lack of luminosity of the flame having anything to do with fuel efficiency, I doubt that such is the case. Table 6 shows that with gas only 41.9 per cent of the heat units in the fuel was found available for use in the smelting chamber as compared with 44.7 per cent for oil and 46.7 per cent for pulverized coal. This might be attributed by some to the nonluminosity of the gas flame, but look for a moment at the flue-gas analysis in the same table. The difference in efficiency is due entirely to the fact that more heat is carried out of the smelting chamber in the waste gases when natural gas is being used than when oil or pulverized coal is used. To this, and not to the nonluminous character of the natural gas flame, I would be inclined to attribute the efficiencies obtained.

In the matter of chrome brick for lining reverberatory copper-refining furnaces my statement was that we discontinued the use of chrome brick because of excessive spalling. This was a mere statement of fact and in no way was intended to imply that chrome brick always spalled or that it is unsuitable for use in a copper-refining furnace. We have used chrome brick as a parting course between silica and magnesite brick and recently placed a small patch in a furnace side wall without experiencing difficulty with spalling. It should be remembered that chrome brick during the World War was not as satisfactory a product as is being placed on the market today. The objection raised to chrome brick by Mr. Pyne is a valid one, however, and should be given consideration.

The Fire Refinery of British Copper Refiners, Limited

By C. H. Aldrich,* Carteret, N. J.

(New York Meeting, February, 1934)

For many years the City of Prescot, about 8 miles northeast of Liverpool, has been the home of British Insulated Cables, Ltd., one of the largest wire mills and manufacturers of electrical equipment in England. Since this location is close to the industrial center of the country and the seaboard, with good railroad transportation facilities, it was a logical site for the new refinery.

Early in April, 1932, the writer was called upon to prepare a preliminary layout and estimate of cost of the proposed plant, and shortly afterward, following a discussion of these figures with officials of British Insulated Cables, in England, the decision was reached to proceed immediately with construction of the plant.

The refinery was built by British Insulated Cables, Ltd. Designing of the plant was started early in May, 1932, orders for equipment being placed as rapidly as possible. On July 18 the first steel was erected and the first charge was cast on Jan. 5, 1933. No difficulties of any consequence were experienced in starting the plant and the grade of copper produced was quite satisfactory after the first charge.

The initial capacity desired was 2500 tons of refined copper per month. Since only one furnace was to be built, it was necessary that it should have sufficient reserve capacity to catch up after unavoidable shutdowns. It was also necessary to lay out the plant so that it would operate efficiently at this tonnage and at the same time permit expansion in the future, if necessary, to several times its original capacity. In order to meet these conditions and also to take full advantage of the space available, the layout shown in Fig. 1 was adopted. The principal plant structures are a blister storage aisle, casting building, wire-bar storage building and a coal-pulverizing building. Additional smaller structures include a brick and charcoal shed, pumphouse, drill room, water-cooling tower, clay-preparation shed, substation and plant office.

Incoming blister is received in either broad-gage railway cars or steam motor lorries in the blister storage aisle, unloaded by overhead electric crane to narrow-gage cars, weighed and sampled, then either sent directly to the furnace or placed in storage. After refining, the copper is cast on the wheel, passes through the cooling bosh and is

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delivered to the wire-bar storage building, where the bars are inspected, weighed and loaded for shipment or stored. It will thus be seen that a straight line flow of material through the plant with a minimum amount

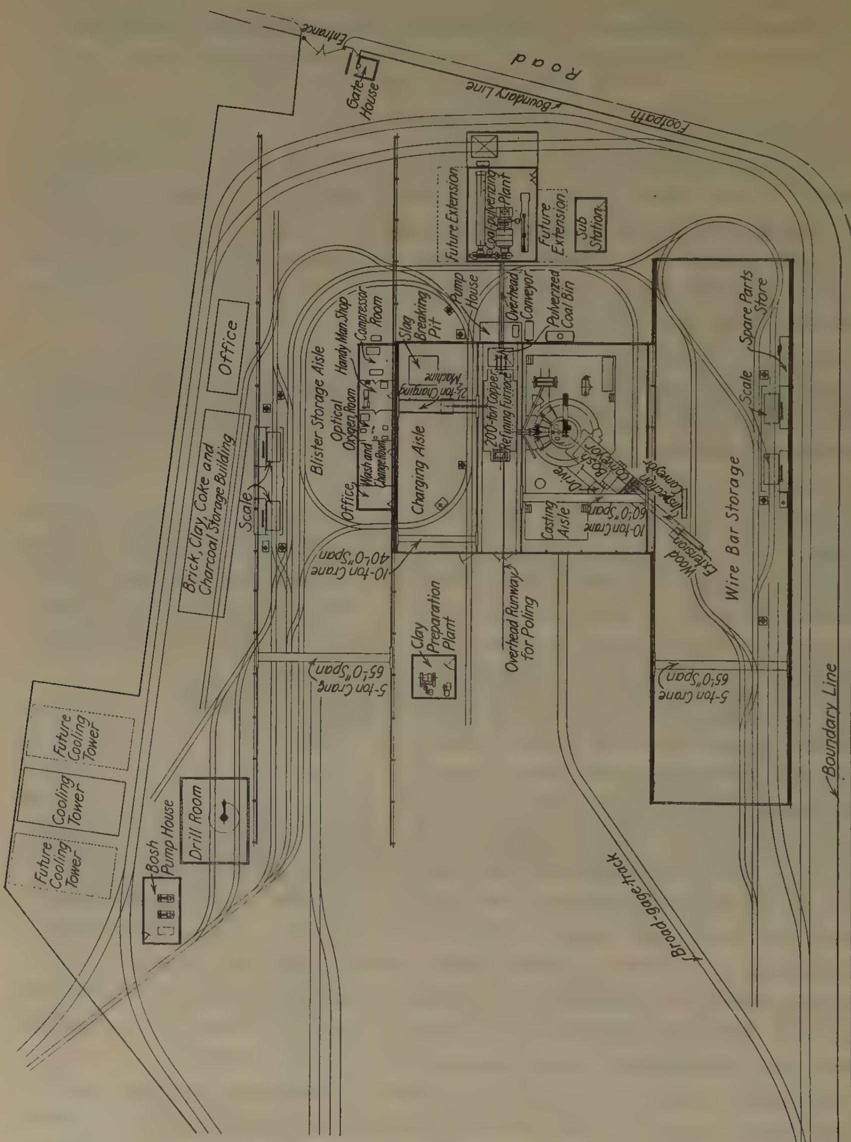

FIG. 1.—GENERAL PLANT LAYOUT, BRITISH COPPER REFINERS CO., LTD.

of handling is secured; also that buildings and storage aisles may be easily extended to permit the addition of two or more extra furnaces without impairing the efficiency of operation.

Blister Storage Aisle

The blister storage aisle is 65 ft. wide by 340 ft. long and is spanned by a 5-ton overhead crane. This aisle contains narrow-gage tracks for car storage and handling of unrefined material, duplicate track scales for weighing and space for ground storage. Broad-gage tracks also run under the craneway at each end, thus providing facilities for the transfer of incoming blister from broad-gage to narrow-gage cars. The drill room in which blister is sampled is conveniently located so that cars that are to be sampled can be passed through it easily and returned to the storage tracks under the crane runway.

Incoming blister is transferred from the docks in Liverpool to the plant either by rail or by steam motor lorries. In the former case it is shipped in open railroad cars from which it can be lifted out by the overhead

Fig. 2.—Main entrance and blister storage aisle.

crane and loaded on to narrow-gage cars for weighing and sampling, then transferred directly to the furnace or placed in storage. The same procedure is followed in case of motor transportation, with the added advantage that the lorry can be placed at any point under the crane runway that may be most convenient for unloading.

For weighing, duplicate 10-ton track scales were installed. These scales are of the highest accuracy obtainable and will show a variation of less than 2 lb. at full load. A test car and weights were provided so that frequent checks of the scale can be made while weighing. Both scales and platforms are protected against the weather by substantial enclosures. Cars are moved over the scales by means of an electric winch, with conveniently located idlers.

The drill room is equipped with a radial drill, cone grinder, drying oven and all facilities for screening and making up samples. It also includes office space and a small laboratory where control assays can be made. The radial drill is in the center of the main drill room, with a narrow-gage track running through on either side. Between the tracks,

on either side of the drill, are two steel drilling tables with 4-in. pins set into their tops, on which the pigs are placed for drilling. This makes it possible to place a shallow pan underneath a pig to catch the drillings.

Blister pigs are sampled by checkerboard drilling in the following manner. Narrow-gage cars carrying the pigs to be drilled are run into the drill room on one side of the drill and the pigs are lifted off by tongs carried on pneumatic hoists supported by small jib cranes. They are spread out on the drill tables, first on one side of the drill then on the other. The drill has sufficient span to cover all the pigs on a table, so that all points to be drilled can be reached easily. After drilling, the pigs are transferred to cars on the outgoing track. Each table in turn

FIG. 3.—FURNACE FROM CHARGING AISLE.

is thus emptied and filled while the pigs on the other are being drilled. For drilling, a 17⁄32-in. drill is used and the drillings are ground and made up for assay in the usual manner.

CASTING BUILDING

The casting building is 100 by 122 ft. in dimensions. It contains three main aisles: the charging aisle, containing the charging and service cranes and slag-crushing pit; the furnace aisle, in which are the furnace, poling and skimming equipment and waste-heat boiler and the casting aisle containing the wheel, cooling bosh and conveyor, all auxiliary casting equipment and the 10-ton casting-aisle crane.

The charging aisle is 40 ft. wide. In this aisle, in addition to regular charges, all materials required in the operation of the furnace, such as brick, clay, charcoal, coke, etc. are handled. Part of the available space is also used for breaking and sorting slag preparatory to outside shipment.

The charging machine is of 2½ tons capacity. It is of the overhead crane type and is similar in design to those used in several of the refineries in the United States. Cars are moved in front of this crane during charging by means of an electric winch. On an upper runway over the charging crane is a 10-ton service crane. This arrangement enables these cranes to move independently of each other, which is a great convenience, especially while there are no spare cranes.

At one end of the charging aisle is the slag-crushing pit, which is of heavy concrete with sectional cast-iron top reinforcement. Its dimensions are 14 by 14 ft. by 2 ft. deep and in it the 4 by 4-ft. cakes of slag are broken up by means of a cast-iron ball, which is lifted and released by a magnet carried on the service crane.

The furnace aisle is 18 ft. wide and contains the furnace on the ground floor and the waste-heat boiler, economizer, induced-draft fan and coal-feeding equipment on the boiler platform. The furnace was designed for a capacity of 200 tons per charge. It was also constructed with an adjustable skimming plate, so that charges of from 100 tons up can be handled easily.

The furnace is set on 4-ft. concrete piers to give good underneath cooling. The bottom is carried on 2-in. thick ribbed cast-iron plates separated from each other by 1-in. spaces to provide for expansion. The tops of the piers were capped with ¼-in. steel plates, a thin layer of graphite being spread between the steel and cast iron. This procedure eliminated all binding in the bottom plates during heating up, the furnace expanding smoothly and evenly.

As a precaution against the solidification of copper around the piers, which sometimes takes place when small leaks occur, 12-in. high firebrick walls were laid between the piers transversely across the bottom of the pit in which the furnace is set. Any copper that may run into these spaces can be easily removed from the sides.

The side plates extending to about 6 in. above the metal line are made of 1-in. steel reinforced with horizontal I-beams between buckstays. Unusually heavy buckstays and tierods were used, the latter, both top and bottom, being equipped with springs.

Water-cooled skewbacks were generally used for supporting the roof in all places that might be exposed when the brick lining burned away. These skewbacks were made of ½-in. boiler plate, all joints being welded. The stiffness required to carry the thrust of the roof arch was secured by the use of plenty of interior bracing.

The narrow space between the side charging doors, where difficulty in maintaining the brick lining is usually experienced, was filled by water-cooled copper blocks lined with clay brick. These blocks were cast of refined copper. They are about 4 in. thick and provided with a deep

flange on each side. Water passages were drilled through the 4-in. sections and the brick lining was wedged in between the flanges.

In lining the furnace, the bottom was shaped by a layer of concrete composed of ganister and cement, tamped in, using the least water possible. Over this were laid $\frac{1}{16}$-in. sheets of steel to keep moisture out of the bottom bricks and the silica-brick bottom was laid on these sheets.

The working bottom consists of two 12-in. inverted arches of silica brick laid up dry with broken joints. Both arches were carefully laid and keyed up, so that if at any time a section of the top layer becomes thin and floats up the lower layer may be depended on to hold the charge. In laying the bottom, any bricks that were warped or off size were discarded and all unavoidable small openings between bricks was carefully filled with hot pulverized silica.

The side walls to about 6 in. above the metal line are built of 15-in. magnesite brick. In order to avoid undermining by the corrosive action of the slag, they were started on the lower bottom course. All magnesite bricks were fitted accurately and laid up using a thin dip of magnesite cement and boiled linseed oil. Below the metal line, expansion joints of $\frac{1}{4}$ in. and above, where exposed to more intense heat, $\frac{1}{2}$ in. in 18 in. were used. The upper part of the side walls was built of 15-in. clay bricks laid as closely as possible, using a thin dip of fireclay. The roof was constructed of 15-in. silica bricks, laid dry and with broken joints.

The furnace is fired by pulverized coal, three Fuller Lehigh burners being used. The use of three burners was favored because this number gives the greatest possible flexibility in the amount of heat supplied and in the application of heat to any part of the furnace. The pulverized-coal storage bin is over the burners on the waste-heat boiler platform and is of 20 tons capacity. The coal feeders are under this bin. Each feeder delivers to a corresponding burner and is driven through a silent chain by a variable-speed direct-current motor. Primary and secondary air fans are also on the boiler platform. They are driven by variable-speed d.c. motors and there is a spare for each service. Adjustable dampers were placed in all air lines, by which the amount of air delivered to the burners may be regulated but the regulation ordinarily used is secured by variation of speed of the fan motors. In order to avoid blocking of the coal line in case a fuse blows on the primary air fan, the feeder motors are electrically interlocked with the former so that they cannot run unless the fan is in operation. All controls for fans and coal feeders are on the ground floor close to the skimming door of the furnace.

The waste-heat boiler is a Babcock & Wilcox straight-tube type with 4264 sq. ft. heating surface. It is set in a welded pan of $\frac{1}{2}$-in. steel plates, on the boiler platform so that water escaping from any leaks will not run down on to the furnace. As originally supplied, this boiler was of steel-

incased construction with asbestos insulation, but because it is sometimes necessary to run with little or no draft it was found necessary to replace the steel casing around the first pass with firebrick. Diamond soot blowers are used for removing dust from the tubes and in addition there are facilities for hand lancing when necessary.

Upon leaving the boiler the hot gases pass through a 1702-sq. ft. high-velocity Green economizer in which their temperature is reduced

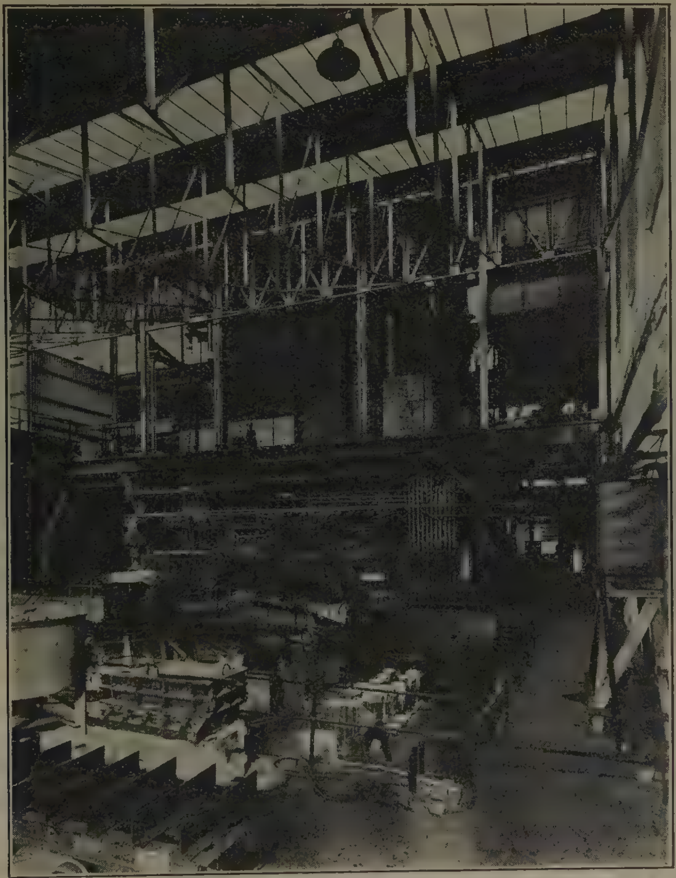

Fig. 4.—Casting aisle, showing mold press, furnace and waste-heat boiler.

to a maximum of 450° F., after which they pass through the induced-draft fan and out of the stack.

The boiler is designed for a working pressure of 250 lb. and 150° superheat but is operating at present at the normal plant pressure of about 150 lb. Steam is piped to the British Insulated Cables power house, where it is used for the generation of electric power and general plant uses. Feed water is supplied by means of a vertical steam-driven

pump and an electrically driven centrifugal pump. These pumps, together with the centrifugal pump supplying hydraulic pressure for operation of furnace doors and mold press, are in a small pumphouse built in the form of a lean-to at the end of the casting building, opposite the furnace aisle. A Copes feed-water regulator is employed to control the water supply to the boiler.

The casting aisle is 60 ft. wide and spanned by a 10-ton crane. It contains the casting wheel, bosh conveyor, mold press and other equipment required for mechanical casting of the regular and side casting of

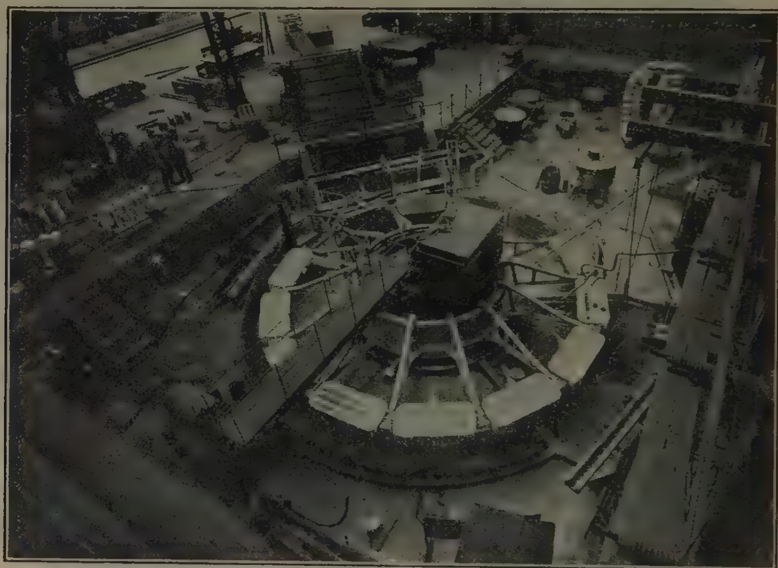

FIG. 5.—CASTING AISLE FROM BOILER PLATFORM.

special shapes; this equipment being on the casting-floor level, which is 4 ft. 6 in. below the level of the ground floor.

The Walker type of casting wheel was selected because of its simplicity and adaptability to the present and future uses for which it is required. It is 26 ft. 10 in. in diameter at the mold centers. The ring is made of cast iron and arms of cast steel. In its construction the highest degree of accuracy in machining was insisted upon in order to insure its running without any vibration. As a further insurance of smooth operation as well as to get the speeds that are required in casting the various shapes, a Variable Speed Gear oil drive was installed. The *A* or oil-pumping unit of this drive is on a concrete foundation at the cooling side of the wheel. The *B* or driving unit is mounted on the wheel instead of the motor, which ordinarily is employed for driving. The operator controls the movement of the wheel by a lever in his cab. Since the *A* unit control is 20 ft. or more from the operator's lever, it was necessary to

take every precaution to avoid any looseness in the linkage connecting them. For this reason the main member of this linkage is a heavy, steel shaft mounted on ball bearings and provided with cranks at the ends which are equipped with adjustable ball and socket joints, one end being thus connected with the *A* unit control, the other, by a short rod, with the operator's lever. The motion is thus transmitted mainly torsionally so that any temperature changes that may take place in the apparatus do not affect the control. As a further precaution this shaft is carefully insulated from the heat given off by the copper in the molds over which it passes. By simple movements of the operating lever it is possible to start and stop the wheel without jar, to get any rate of acceleration or deceleration desired and to run forward or backward at any speed up to the maximum of which the drive is capable.

The operator's cab is on the center of the wheel directly in front of the pouring ladle. In addition to the control lever for the wheel he has the controller for the electrically driven ladle hoist or tilter and an arrangement by means of which he can move the ladle forward or back or tilt it sideways while pouring.

Initially, three pocket copper wire-bar molds were installed. These molds are large enough to handle the capacity of the furnace in a reasonable time and can be easily enlarged to cast four bars each if necessary. The first set of molds was shipped from the United States.

While casting is in progress the molds are continuously cooled by water sprays set underneath. These sprays are made up of two 4-in. pipes bent to the same radius as the mold circle and each perforated with three rows of ¼-in. holes drilled 3 in. apart and staggered. Sufficient water pressure is used so that the bottoms of the molds are well covered but not enough so that the water reaches the top surface of the molds. Underneath spraying starts as soon as the molds pass out from over the casting pit. When they dump into the bosh they are sprayed on both sides to reduce the temperature quickly to the proper degree for painting. In principle the idea is to maintain the molds at a fairly uniform temperature rather than to allow them to become excessively hot, then cool quickly by heavy spraying.

The mold painter sits in a shallow pit and sprays the molds with an air-operated spray gun as they pass before him. In order to enable him to keep the molds at exactly the proper temperature, the valves controlling the final sprays are located so as to be within easy reach.

The molds are dumped and righted by means of roller-bearing dumping fingers running on a substantial guide rail. At the point of dumping the jar is minimized by an adjustable spring-cushioned arm that engages the dumping finger as the mold turns over. As a further precaution against transmission of jar to the wheel, the dumping roll on which the mold rides at the finish of the dump is carried on an I-beam spanning the

bosh, which is entirely separate from the one to which the guide bars are attached.

The molds dump at a point 220° from the pouring ladle, the bars falling into the cooling bosh, which is of ample proportions to give sufficient cooling. As they drop from the molds they strike an inclined baffle plate and slide down on to the bosh conveyor, which carries them a distance of 6 ft. horizontally through the water before the inclined section is reached. This conveyor is of very substantial construction and of the pan type; a type that our experience has shown to be least likely to become clogged by scrap or heavy loads and to be most economical of power. By means of a variable-speed d.c. motor drive, the conveyor may be run at from 8 to 24 ft. per minute. The final temperature of the bars is regulated by a set of sprays mounted over the inclined end of the conveyor, the valves being conveniently reached from a bridge that crosses the conveyor at ground-floor level.

Cooled bars drop from the upper end of the bosh conveyor on to an inclined steel plate apron, which delivers them to an inspection conveyor 27 ft. long, on which they are gaged, inspected and chipped. The speed of this conveyor, which is independently driven, also may be varied from 9 to 27 ft. per minute, thus giving ample flexibility for handling the various shapes that may be produced.

The mold press is of substantial construction and hydraulically operated. It consists of a cast-iron frame which supports the movable core plate and cores, spanning a heavy copper bottom plate supported on a carriage operating on a short section of track set flush with the floor. Timken bearings are used on the wheels of this carriage, so that despite the heavy load it carries it can be moved easily. Cast-iron end blocks and side fixtures are clamped together on the bottom plate and molten copper is tapped directly to the mold press, through a launder. After the mold has set the cores are raised, the carriage pushed out from under the press and the mold lifted off by means of a pair of heavy tongs attached to the hook of the overhead crane. The water-cooled wire-bar cores are made of refined copper, cast hollow and accurately machined all over. The first set was made in the United States but all cores subsequently required have been successfully produced at Prescot.

In addition to the equipment mentioned, the casting floor is equipped with molds for billets and cakes, bull ladles and an oil-fired portable holding furnace, all of which are used in the side casting of these shapes. Space is also provided on this floor for the storage of spare wire-bar, ingot and ingot-bar molds.

Wire-bar Storage

The wire-bar storage building, which is 65 ft. wide by 260 ft. long, is totally enclosed so that inspection, weighing and shipping operations

can be carried on at all times without interruptions resulting from weather conditions. It contains a 5-ton overhead crane, inspection tables, duplicate 10-ton track scales similar to those provided for the weighing of incoming blister, narrow-gage and broad-gage tracks for loading outgoing shipments, and space for ground storage of refined copper.

Wire-bars and other shapes cast on the wheel are delivered directly into the wire-bar storage building by the inspection conveyor so that no narrow-gage transportation is required at this point. From the conveyor they are either lifted and placed on narrow-gage cars by means of an air-lift carried on a jib crane or, more often, piled by the overhead crane on a table at the end of the inspection conveyor and lifted in loads of 5 tons each by the crane and transferred to cars in front of the scales on which they are weighed. They may then be loaded directly into open railway cars for shipment or piled for stocking. Practically all loading and unloading of copper in this building can be done with the overhead crane, which makes this handling an inexpensive operation.

Cooling System

Water for all plant operations is obtained from the Liverpool supply. For cooling molds, wire bars, jackets, etc. a very large flow of water is required, particularly during the casting period. On account of local conditions it was necessary to use equipment that would require little space but that would handle a large volume of water and cool it quickly. To meet these conditions a Visco cooling tower, designed to dissipate the maximum heat units that would be produced while casting, was installed. It consists of a rectangular wood tower or chimney about 75 ft. high with an enlarged lower section where the incoming hot water is broken up by dropping over a system of launders and wooden strips. Cool air, which is drawn in through louvers at the bottom by natural draft, absorbs the heat of the dropping water as it passes up through the tower and out of the top. A concrete tank, 50 ft. long by 26 ft. wide by 6 ft. deep, underneath the tower serves as a reservoir and this is kept at a constant level by means of a float valve attached to the incoming supply. One cooling tower has sufficient capacity for present operations. To take care of possible future expansion, space was reserved for two more.

The pumphouse is adjacent to the cooling tower, the pumps being set at the level of the bottom of the reservoir so that they are self priming. Two centrifugal pumps, with a capacity of 2500 gal. per minute at 100-ft. head were provided, and space was allowed for a third. At present, one pump has ample capacity for all plant requirements, the second serving as a spare.

In the casting building, cooling jackets, sprays and the cooling bosh drain into a deep sump of generous proportions. Outside this sump and

on a level with the bottom are two vertical-shaft centrifugal pumps, each having a capacity of 2500 gal. per minute at 75-ft. head. These pumps were built with extended shafts, the motors being on floor level. They are started and stopped by float-operated controllers so that they run intermittently; starting when the sump tank reaches a predetermined level and stopping when it is empty. In this case again one pump can easily handle the total amount of water required, the second serving as a spare.

Thus all water returned from the plant passes through the cooling tower and the rate of cooling is proportional to the rate of circulation at

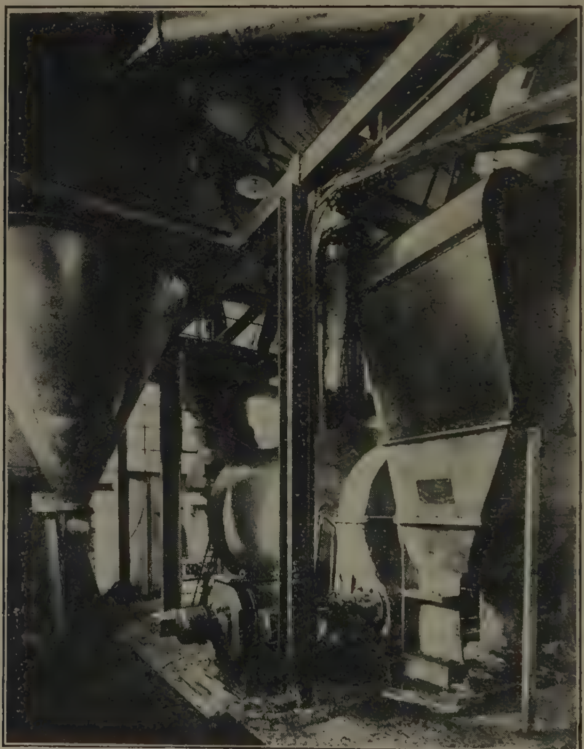

Fig. 6.—Coal-pulverizing plant.

all times. This system has given quite satisfactory results in practice. Should additional cooling be required it can be obtained easily by auxiliary circulation over the tower.

Coal-pulverizing Plant

For the preparation of the pulverized coal required by the refining furnace, a plant containing some novel features was installed. This plant has a rated capacity of 5 tons per hour pulverized to a fineness of

85 per cent through 200 mesh. In its design the principal considerations were to secure the greatest possible reliability and freedom from shutdowns for repairs together with the lowest cost of operation. This was particularly necessary because the relatively small tonnage requirements did not justify the installation of spare equipment.

Low-sulfur coal is received in 10-ton four-wheel railway cars and dumped through a grizzly into a track hopper, by raising one end of the car with a hydraulic ram and allowing the coal to slide out of the other. It is drawn from the bottom of the hopper and delivered to the top of a 50-ton raw-coal storage bin by a Redler conveyor having a capacity of 20 tons per hour. A disk feeder draws coal from the storage bin and feeds it to a coke-fired Ruggles-Coles drier equipped with automatic temperature control. Waste gases are drawn from the drier by a fan, passed through a small cyclone and out through a superimposed stack containing water sprays to remove the last of the dust. In passing, it is interesting to note that this type of dust-removal equipment appears to work much more effectively on English than on American coals.

Dry coal passes into a Redler conveyor which elevates it to a small surge bin over the mill. For pulverizing, a Johnson air-swept ball mill is used. The outlet end of this mill is connected to a separator shown in Fig. 6, which separates the oversize and returns it to the mill by means of a helical conveyor in the hollow outlet shaft. Adjustable vanes in this separator regulate the fineness of the product.

The finely pulverized coal together with circulating air is drawn from the separator and passed through a large cyclone which removes the coal; the air is piped back to the inlet end of the mill. Pulverized coal is discharged from the cyclone through a rotating air lock into a continuously running Redler conveyor, which delivers it to the 20-ton storage bin on the boiler floor, over the furnace burners. The plant is arranged so that the conveyor that delivers coal to the furnace bin can be easily extended to take care of additional furnaces and, if required, the present capacity of the pulverizing plant can be doubled by the installation of additional equipment.

One of the most interesting features of this plant is the use of Redler conveyors for all coal handling, including transport lines to the furnace bins, for which air usually is employed. The results obtained in every case have been quite satisfactory.

Transportation

In order to secure the greatest possible flexibility in broad-gage railway transportation, a loop track was run completely around the plant so that materials may be moved in or out in either direction. In addition to this loop, spur tracks were run in to the blister storage aisle, furnace-pole storage at the end of the casting aisle and wire-bar storage building.

Materials are handled within the plant by a narrow-gage system. The gage used is 30 in. and tracks are laid so that no curve has a radius of less than 35 ft. Most of the cars are typical billet cars equipped with cross T-irons so that loads can be readily picked up by the charging crane. These cars have a rated capacity of 10 tons each and are equipped with double trucks, Timken bearings and automatic couplers. In addition to the billet cars a number of side-dump hopper cars for handling loose materials were included.

For motive power two light Diesel locomotives were supplied.

Electric Power

Electric power is received from two sources. The principal supply is purchased from the local power company. This enters the plant at 6000 volts, three-phase, 50 cycles and is stepped down to 240 volts in a substation near the coal-pulverizing plant. Direct current is received from the British Insulated Cables' power plant for use with motors where variable speed or high starting torque is essential. It is used on: (1) primary and secondary air fans, (2) induced-draft fan, (3) pulverized-coal feeders, (4) bosh and inspection conveyors, (5) ladle-tilting hoist, (6) charging crane. All other equipment is operated by alternating-current motors.

Auxiliary Equipment

A lean-to built alongside the casting aisle contains the air-compressor room, handy-man's shop, optical oxygen room, wash and change room and casting-building office.

For supplying high-pressure air a direct-driven air compressor with a capacity of 300 cu. ft. of free air per min. at a pressure of 90 lb. per sq. in. was installed. Low-pressure air at about 18 lb. per sq. in. for blowing the copper in the refining operation is supplied by a 500-cu. ft. Nash "Hytor."

The handy-man's shop is equipped with a forge and blacksmith's tools, drill press, grinder and pipe-threading machine. Repairs to rabbles, bars and inspectors' tools, the fitting up of pipes for blowing and general simple plant repairs are handled in this shop.

The optical oxygen room contains a large saw for cutting samples from wire bars and cakes; a small, high-speed saw for cutting small samples; polishing and etching equipment, and metallographic microscope.

For the storage of firebrick, clay, cement, charcoal, etc. a building 20 ft. wide by 100 ft. long was constructed beside the blister storage aisle. Supplies are brought to this building over a broad-gage track running along one side and are unloaded from cars through a number of doors on the car-floor level. A narrow-gage track running through the building

on the opposite side is used in making deliveries to the plant as the materials are required.

Buildings

All building walls are constructed of precast concrete blocks made in the British Insulated Cables' concrete-block plant. Corrugated asbestos was used for the roofs and a generous proportion of corrugated, reinforced glass in these roofs gives exceptionally good lighting. Good ventilation of the casting building is secured by the use of a large monitor over the boiler aisle. This is particularly valuable because it enables the large volume of steam given off from the wheel and bosh during casting to escape quickly. Additional lighting and ventilation is secured by the installation of plenty of movable steel-sash windows. All floors in the buildings are concrete and nearly all of the outside working space throughout the plant is concreted.

Furnace Operation

The fire refining of Roan Antelope blister did not present any real difficulties. The method used is as follows: Four or five wheelbarrows of sand are first thrown over the bottom of the furnace, a thin layer of clean scrap wire from the wire mill is put in as a cushion and the blister is charged with the charging crane in the usual manner. The charge is melted and when all off the bottom is skimmed clean. Air pipes are put in and the charge blown and flapped until a say ladle button shows a large block grain and optical examination indicates a content of 0.90 per cent oxygen. When this point is reached, the bath is skimmed clean, and a fairly thick layer of low-sulfur coke is added. Several intermediate light skimmings usually are required during the oxidizing period. It is usually possible to add a "coking charge" of from 8 to 10 tons of rejected wire bars or refined scrap at this point if desired.

Poling is carried out in the usual manner, a good grade of green, hardwood poles being used, and is continued until a small block sample cast with a hand ladle in a cast-iron mold shows a fine-grained full set and the absence of a "spot." When this point is reached the optical oxygen determination will indicate about 0.03 to 0.04 per cent oxygen and casting may be started. During casting the set is watched carefully and held at the desired level by careful temperature control, and the occasional application of a pole when necessary. The bars produced are fully equal in appearance to the best electrolytic wire bars, and in electrical and mechanical properties show an appreciable margin over the requirements of all standard specifications for electrolytic copper.

The slag produced runs from 2 to 3 per cent of the material charged. It is skimmed into large boxes measuring 4 by 4 ft. by 18 in. deep, made

of ¾-in. welded steel plates. These boxes are removed from the skimming door by the charging crane and after cooling the slag is broken up, the metallics picked out and shipped to an associated plant for further treatment.

Acknowledgment

The writer wishes to express his thanks to Mr. G. H. Nisbett, Managing Director of British Insulated Cables, Ltd., for permission for the publication of this article, and to Mr. D. W. Aldridge, Works Manager of British Copper Refiners, Ltd., for assistance given in its preparation.

Removal of Arsenic and Antimony from Copper by Furnace-refining Methods

By W. J. Hillenbrand,* R. K. Poull* and H. C. Kenny,* Hubbell, Mich.

(New York Meeting, February, 1934)

The soda-ash method for eliminating arsenic from a molten copper bath has been described previously in some detail.[1] Briefly, the process consists in blowing powdered sodium carbonate under the surface of the partially oxidized melt and in removing the slag formed, which will contain a certain proportion of the arsenic formerly in the bath. Sodium carbonate melts readily at the temperature of molten copper, and when molten it dissolves arsenic oxide rather rapidly. In the process, sodium arsenate is formed which is removed along with the excess of molten soda ash. In order that arsenic may be eliminated in this way, it must first be oxidized to the pentavalent condition. This, however, is easily accomplished by blowing air into the molten copper until a fair excess of cuprous oxide is present.

The manner in which soda ash is introduced into the furnace has a considerable effect upon the efficiency of the process. In the earliest attempts at arsenic removal, soda ash—often in combination with slaked lime, for sodium carbonate is not the only solvent that will remove arsenic—was thrown upon the surface of the bath, was allowed to melt, and was removed after it had dissolved a sufficient amount of arsenic. At present powdered soda ash is generally blown in under the surface of the melt with the aid of compressed air. Melting immediately as it comes into contact with the molten copper, it can react with considerably more of the bath than by the former method, and thus can remove arsenic more rapidly and more completely. Since the resulting slag is removed at once, its very destructive effect upon the furnace refractories is minimized.

The effectiveness of the soda-ash method may be gathered from the results of the following test. A 900-lb. charge of copper was melted in a small, oil-fired reverberatory furnace, and sufficient metallic arsenic was added to the bath to bring the concentration of that element to about 0.5 per cent. Air was blown into the melt until its oxygen content was just under 1 per cent by weight. The bath was then sampled and soda ash was blown in under the surface in 5-lb. batches. After each addition

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[1] C. T. Eddy: Arsenic Elimination in the Reverberatory Refining of Native Copper. *Trans.* A.I.M.E. (1931) **96,** 104.

of soda ash the slag was removed and a sample of the bath was taken. Fig. 1 illustrates the rapidity and thoroughness with which the arsenic was removed from the copper.

By treatment with soda ash many other elements besides arsenic may be removed from a copper bath.

In the course of an investigation upon which the authors were engaged, it became necessary to determine the effect of soda ash upon both arsenic and antimony when those elements were present together in a bath of copper. The situation arose in the following manner.

Analyses of refined Lake copper, when performed in the refinery laboratories, have failed to disclose the presence of antimony. Through

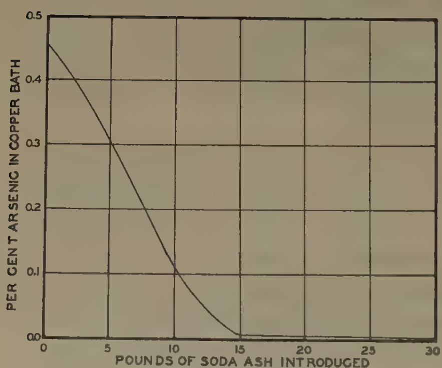

FIG. 1.

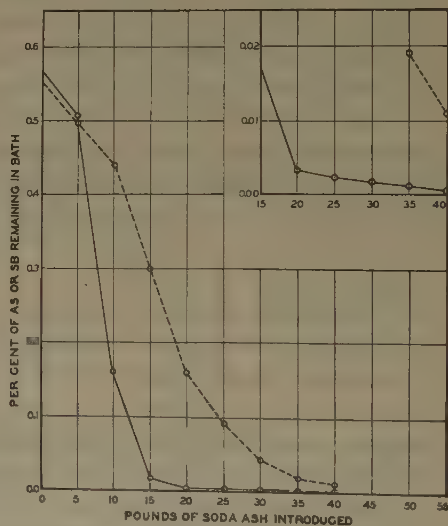

FIG. 2.

FIG. 1.—ILLUSTRATING RAPIDITY AND THOROUGHNESS OF REMOVAL OF ARSENIC FROM COPPER.

FIG. 2.—RESULTS OF TESTS OF ARSENIC AND ANTIMONY ELIMINATION. ——— Per cent As; .... Per cent Sb.

the use of an accurate gravimetric method in which 1000-gram samples of copper were taken for analysis, it was demonstrated by the authors that antimony is never present in amounts as great as 0.0001 per cent, a result which was confirmed by Prof. Bart Park, who worked with a spectroscopic method.[2] Analyses of Lake copper from other laboratories, however, have generally reported small amounts of antimony. Furthermore, antimony minerals are reported as having been found in the Lake Superior district, although a careful mineralogical and chemical search

[2] B. Park and E. J. Lewis: Estimation of Small Amounts of Antimony in Copper. *Ind. & Eng. Chem.*, Anal. Ed. (1933) **5**, 182.

conducted in connection with this investigation failed to detect antimony except in the upper levels of one of the mines long since abandoned.

The discrepancies in the chemical analyses of the refined copper could, of course, be attributed to the inadequacy of routine methods of copper analysis where traces of foreign elements are involved, but it was thought that added evidence as to the absence of antimony would be highly desirable.

All Lake copper—with the exception of the small tonnage of arsenical copper produced—is subjected to a soda-ash treatment in the process of refining. If antimony was present in the copper before the soda ash was added, it should be removed by that solvent at least in part and should thus be present in the soda slag in a greater concentration. Consequently, samples of soda slag were taken at intervals during the removal of the arsenic from a 250-ton furnace charge and were combined so as to

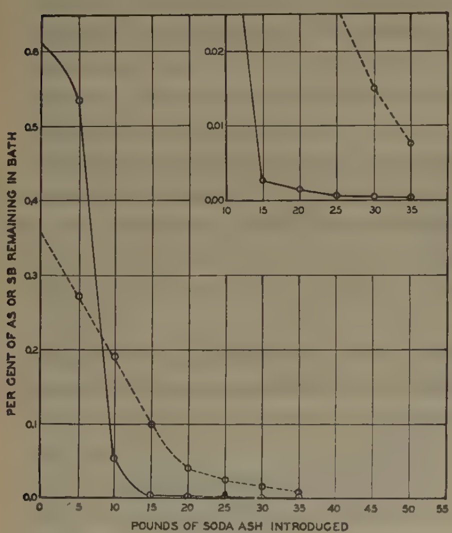

Fig. 3.

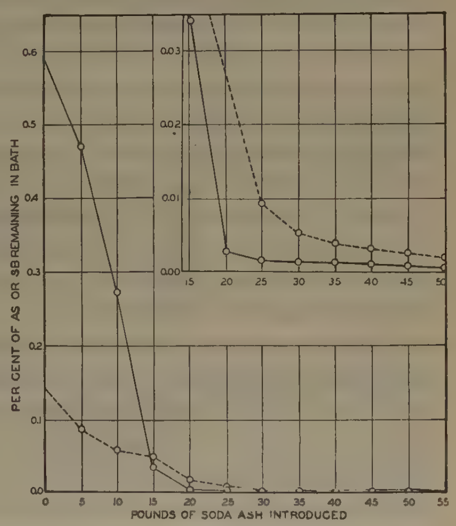

Fig. 4.

Figs. 3 and 4.—Results of tests of arsenic and antimony elimination. ——— Per cent As; .... Per cent Sb.

represent the slag resulting from three successive one-ton lots of soda ash. These three samples were analyzed for arsenic and antimony with the following results:

	No. 1	No. 2	No. 3
Per cent arsenic	5.25	6.02	2.66
Per cent antimony	0.0033	0.0069	0.0027
Ratio arsenic to antimony	1600:1	870:1	980:1

The charge of copper before soda-ash treatment contained nearly 0.1 per cent arsenic. After treatment 0.0016 per cent remained. Very

nearly 500 lb. of arsenic had been removed by the soda ash and, as calculated from the slag analyses, somewhat less than 8 oz. of antimony. The antimony removed was equivalent to slightly less than 0.0001 per cent of the copper in the furnace.

Doubt as to the completeness of removal of the antimony still remained. In order to clear up this uncertainty several charges of copper of about 1000 lb. each were melted in the small reverberatory furnace previously mentioned, and both arsenic and antimony were added to them in predetermined amounts. The melts were oxidized and were sampled as before; soda was introduced in 5-lb. batches, and the bath was sampled after each addition. Figs. 2, 3 and 4 show the results obtained from three such tests. The percentages of both arsenic and antimony remaining in the bath are plotted against the total amount of soda ash added.

The following generalizations can be drawn from the data obtained. When arsenic and antimony are both present in a copper bath they can be eliminated simultaneously by soda-ash treatment. If the percentages of the two elements are fairly high and nearly equal, the ratio, arsenic eliminated to antimony eliminated, is approximately 6 to 1, arsenic being more readily removed than antimony. As the percentages of the elements decrease, the above ratio decreases also. Complete removal of one of the elements is impossible without the removal of the other.

In no case was a ratio for arsenic removed to antimony removed as great as 1000 to 1 obtained or even approached, hence the authors' original purpose in securing the data was not directly served. However, from the data it can be safely inferred that to obtain such a relationship the relative proportions of arsenic to antimony in the original melt must have been of the same general order, namely 1000 to 1. It follows as a natural conclusion that the 8 oz. of antimony removed from the 250-ton charge of copper before mentioned represented substantially its entire antimony content and that the refined copper finally contained well under 0.00001 per cent of that element, a quantity beyond the reach of routine chemical analysis.

A Review of Work on Gases in Copper

By O. W. Ellis*

(New York Meeting, February, 1932)

Before entering upon a general discussion of the fascinating, but at present rather controversial, subject of gases in copper, the author feels that some attention should be directed to the work which has been carried out on the solubility of gases in metals. The outstanding contributions to our knowledge of this subject have been made by Sieverts and his collaborators[1]† and by Iwasé.[2] Both the German and the Japanese investigators employed apparatus of the same general design in their investigations. Full reference need only be made, therefore, to the German scheme, which is diagrammatically illustrated in Fig. 1.

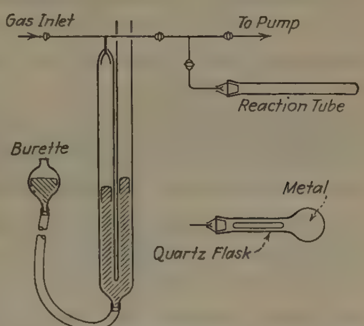

Fig. 1.—Sieverts' apparatus for investigating solubility of gases in metals.

This, to use the words of Sieverts, which he employed in describing his arrangement of apparatus at the 1928 meeting of der Deutschen Gesellschaft für Metallkunde[1a]:

> shows a diagram of the usual mode of procedure. A quartz flask contains the metal (for high temperatures a highly refractory porcelain tube is used) and is connected with the rest of the apparatus by means of a stopcock. The other parts consist of a burette divided into 1/10 c.c., a leveling tube, and an adjustable mercury reservoir. The burette is connected on one side with the gas inlet and on the other side with the absorption flask; the connection can be interrupted by means of suitable taps. Another tap permits the connection of the flask and the Töpler pump. In an experiment the quartz flask is first exhausted and then a measured quantity of gas is let in from the burette. In the first place the volume of the flask when empty is found by calibration with an inert gas. The pressure is read off on the level tube of the burette, and the temperature measured by means of a thermocouple whose welded junction lies close to the outside of the quartz flask, which is heated in an electric furnace. In several experiments the quantity of absorbed gas was transferred to a burette by means of a pump and measured.

Care was taken by Sieverts to cool the ground-in joint between the reaction flask (or tube) and the rest of the system. Suitable precautions

* Ontario Research Foundation.

† Superior figures in parentheses refer to bibliography at end of paper.

appear to have been taken to allow for such diffusion of gas from the reaction flask (or tube) as occurred when the absorption phenomena were being investigated at temperatures in excess of about 1000° C.

Iwasé[(2)] improved on the Sieverts apparatus in two ways. First, by (to all intents and purposes) fusing a short silica tube into the reaction flask, so that the thermocouple used in registering the temperature of the metal under observation was almost completely surrounded by the melt, he increased the accuracy of his measurements of temperature. Second, he immersed all the ground-in joints in his apparatus in mercury, so as to insure that no leakage of gas could occur at these points.

In the fourth paragraph of his paper, Iwasé suggests that Sieverts and his collaborators did not allow the metals and gases to be brought in contact for a sufficient time at each temperature, so that the values given by him do not seem to be those of the saturated ones. "Hence," says Iwasé, "the present writer determined the amount of gases absorbed to the point of saturation by metals by bringing gases and metals in contact for a sufficient time at various temperatures." Despite this statement to the effect that the German investigators "did not allow the metals and gases to be brought in contact for a sufficient time at various temperatures," one finds it hard to conceive how they could have failed in their object of reaching equilibrium in their tests. In Sieverts' paper on Solubility of Hydrogen in Copper, Iron and Nickel, for example, he specifically states that "the control test made at the end of a set of tests did thus not only give a guarantee of the accuracy of the last measurement, but permitted also of determining whether gas losses or gains had taken place; in other words, whether the preceding measurements deserved confidence or not."

Laws of Solubility of Gases in Metals

At the moment it appears that all one can do is to present the results of both schools. This is done here, in so far as copper is concerned, in Figs. 2, 3 and 4, which show the absorption coefficients of the more important gases in copper at different temperatures.

In Fig. 2, the results of Sieverts' and Krumbhaar's individual experiments with hydrogen are represented by the open circles, while Iwasé's are shown by the black dots. The difference between the two sets of results is most striking. In this connection it may be noted that Sieverts found the mass of gas dissolved in copper to be proportional to the square root of the gas pressure, which leads immediately to the view that the molecule of hydrogen in solution in copper contains only one atom. The equilibrium between hydrogen and cuprous oxide in molten copper must, therefore, be represented by the equation

$$2H + Cu_2O \rightleftarrows 2Cu + H_2O.$$

Recent work by Allen[3] lends considerable support to Sieverts' law, as it may well be termed. It will be recalled that the solubility of gases in most liquids obeys Henry's law, which states that the mass of a gas dissolved by a given volume of a liquid at a definite temperature is proportional to the pressure (not to the square root of the pressure) of the gas.

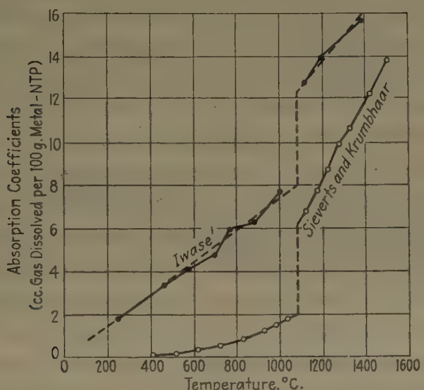

FIG. 2.

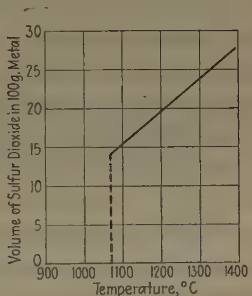

FIG. 3.

FIG. 2.—SOLUBILITY OF HYDROGEN IN COPPER (IWASÉ AND SIEVERTS AND KRUMBHAAR).
FIG. 3.—SOLUBILITY OF SULFUR DIOXIDE IN COPPER (SIEVERTS AND KRUMBHAAR).
[From *Jnl.* Inst. Met. (1930) **43**, 74.]

In Fig. 3 are shown the results of Sieverts and Krumbhaar's work on the solubility of sulfur dioxide in copper. The values are unexpectedly high. Even more surprising is the fact that the solubility of sulfur dioxide in copper is, like that of hydrogen, proportional to the square root of the gas pressure, from which Sieverts argues so late as 1928 that "*there can be no question here of dissociation of the gas into two components.*"[1g] One would have been tempted to explain, as others have done, the somewhat remarkable solubility of sulfur dioxide in copper by assuming that the gas *was* decomposed in solution, with the result that cuprous sulfide and cuprous oxide were formed on freezing in accordance with the equation

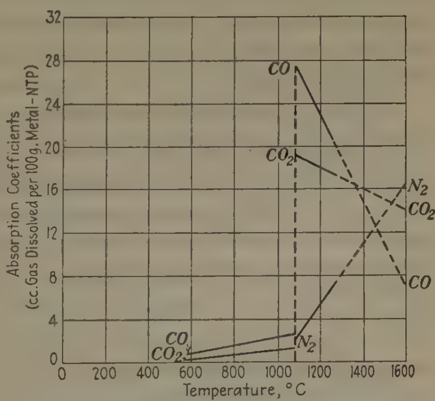

FIG. 4.—SOLUBILITY OF CARBON MONOXIDE, CARBON DIOXIDE AND NITROGEN IN COPPER (IWASÉ).

$$SO_2 + 6Cu \rightleftarrows Cu_2S + 2Cu_2O$$

This was the explanation given by Stubbs[4] in 1913. He showed that

approximately 70 per cent of the sulfur dioxide dissolved in molten copper reacts with the metal to form cuprous sulfide and cuprous oxide. There can be no doubt that this reaction does occur, since microscopic examination under polarized light by Wells and Dalzell(5) of samples of previously deoxidized copper, which had been melted and solidified in vitreosil crucibles in the presence of gaseous sulfur dioxide, revealed both cuprous sulfide and cuprous oxide after this treatment. It is open to question, however, whether the action is reversible, as Stubbs apparently believed, since the particles of cuprous sulfide and cuprous oxide (whether eutectic or proeutectic), which precipitate on freezing from what is, after all, nothing but an alloy of copper, sulfur and oxygen, are separated from one another by a continuous network of copper. They are thus largely, if not entirely, prevented from subsequently reacting to form sulfur dioxide. Wells and Dalzell have argued that porosity would be expected to result each time an alloy of copper, sulfur and oxygen was remelted and resolidified in vacuo if reaction between the sulfide and the oxide could occur. This argument is somewhat difficult to follow, but their observation that such remelting and resolidification can be carried out three times in succession without apparent decrease in the amounts of cuprous sulfide and cuprous oxide, as revealed by microscopic examination, appears to indicate that reaction from left to right does not occur.

To sum up, it may be said that the solubility in copper of sulfur dioxide *per se* has yet to be determined. In fact, we stand in much the same position today as did Hampe(6) who in 1873 had arrived at the conclusion that molten copper absorbs sulfur dioxide, which is retained in part when the metal solidifies, and which is not removed, except perhaps superficially and very slowly, when the metal is heated in a stream of indifferent gas.

In Fig. 4 are shown the results of Iwasé's work on the solubilities of carbon monoxide, carbon dioxide and nitrogen in copper. In view of the fact that Sieverts and his collaborators obtained results in direct contradiction to those of Iwasé, considerable confusion has resulted. Sieverts states categorically that "nitrogen is not dissolved by metals." In the same paper he remarks that he examined carbon dioxide "to ascertain its behavior toward solid and liquid copper. As inferred among others by Hampe carbon dioxide is not soluble in copper. Carbon monoxide was likewise heated with copper up to 1520° C. No solubility could be noticed, contrary to the literature."

In commenting upon his work with the carbonic gases, Iwasé remarks that

> it is said that CO dissociates into C and CO_2 when it is heated with metallic copper at a temperature higher than 550° C. If this is the case, the volume of the gas decreases as the result of the dissociation, and hence the net volume of occluded gas cannot be known. But in the case of the present experiment, *i. e.*, with carbon

monoxide, the gas treated is passed through a layer of copper gauze at red heat for the elimination of the oxygen in it. Hence if the gas dissociates, there should be found some quantity of CO_2 in the purified gas; but this gas was not detected. Hence, the decrease in volume of CO which was in contact with the heated copper powder can be taken as dissolved in the metal.

Before entering upon a discussion of the qualitative investigations of the behavior of gases toward copper, consideration must be given to the work of Allen[3] on the decomposition of steam by molten copper. One hundred and thirty-two years ago, Proust[7] reported that water was decomposed by copper, and numerous investigators since have studied

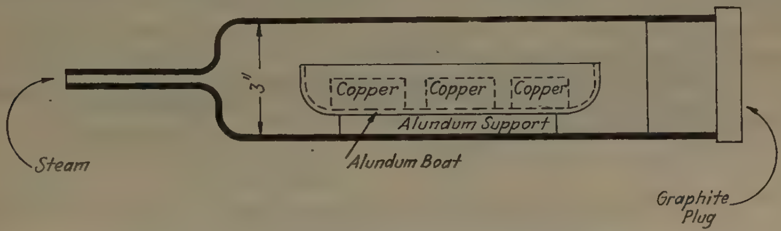

FIG. 5.—ARRANGEMENT OF FURNACE (ELLIS).

the reactions between steam and the solid metal. The immense importance of these reactions, in so far as liquid copper was concerned, has been made abundantly clear by the author.[8] In a very rough experiment

copper was heated, melted, solidified and cooled in an atmosphere of water vapor. The maximum temperature reached was about 1300° C., which was maintained for 20 min. The metal was allowed, after fusion, to cool to a temperature of 500° C., when it was removed from the furnace and immersed in alcohol to prevent surface oxidation. The cooling from 1300° to 500° C. occupied about one hour. The arrangement of the furnace in which the fusion took place is shown in Fig. 5.

Analysis of the ingot showed it to contain 144.9 per cent of water vapor, 49.2 per cent of carbon dioxide, 9.7 per cent of carbon monoxide, and 5.4 per cent of hydrogen, the percentages being in terms of the volume of the copper. The ingot was full of blowholes. It is almost certain that the carbon monoxide and dioxide resulted from reaction of the water vapor with the graphite plug which was used to seal one end of the furnace, since the metal used had been fused in vacuo before it was introduced into the furnace. The copper was free from cuprous oxide before melting but after melting was found to contain oxygen in this form. This oxygen no doubt originated from the dissociation of water vapor ($2H_2O \rightleftarrows 2H_2 + O_2$), the hydrogen entering the metal as gas and the oxygen uniting with the copper to form cuprous oxide, since no oxygen as gas was found in the copper.

Allen, who was endeavoring to discover the source of the type of unsoundness generally found in commercial ingots, had reached the conclusion that hydrogen was intimately, though indirectly, connected with this form of trouble. He was led to the view that the reaction

$$H_2 + Cu_2O \rightleftarrows 2Cu + H_2O$$

was in reality the cause of the unsoundness and conducted a series of experiments similar in principal to that described above, but far more

refined in character. Copper was melted in a fireclay boat, in a closed-end silica tube, arranged as in Fig. 6, and a gas-supply system was made in such a way that the metal could be treated first with hydrogen and then with steam generated from boiled-out, oxygen-free water, without admitting any air into the furnace when the charge was made. Fuller details of Allen's experiments may be omitted here, but the facts that the copper was oxidized as a result of his treatment, that the solidified metal was unsound if cooled rapidly enough, and that hydrogen was produced and could be removed from the metal and measured, may well be noted.

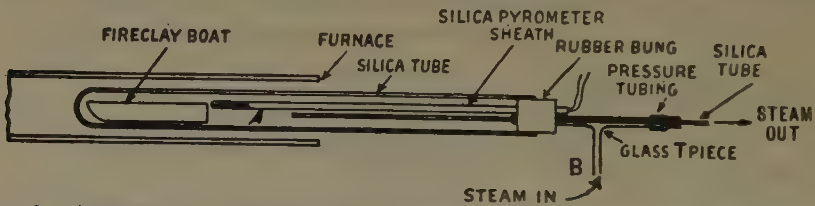

FIG. 6.—ALLEN'S APPARATUS FOR TREATMENT OF MOLTEN COPPER WITH STEAM. (From *Jnl.* Inst. Met. (1930) **43**, 118.)

The velocity of reaction increased with temperature, the amount of hydrogen produced per hour varying from 0.2 c.c. at 1060° C. to 10.6 c.c. at 1275° C. The unsoundness was found to resemble that of commercial copper in the following particulars:

1. It caused a fall in density of the same order.
2. The holes were found associated with the Cu-Cu_2O eutectic in the grain boundaries and between the arms of the copper dendrites.
3. It was produced in the presence of oxygen.
4. It was prevented by very slow cooling during solidification.
5. It was increased by increasing the temperature.

Allen later attempted to measure the equilibrium between molten copper and steam, using the arrangement of apparatus in Fig. 7. The results of four of his experiments are shown in Table 1.

TABLE 1.—*Relation of Oxygen Content to Hydrogen Content of Copper at 1150° C.* (*Allen*)

(Partial Pressure of Water Vapor, 234 Mm.)

Hydrogen Pressure, Cm. Hg	Hydrogen Content (Calculated), Per Cent	Oxygen Content, Per Cent
0.015	0.0000087	0.284
0.067	0.000018	0.155
0.340	0.000091	0.050
6.607	0.00018	0.006

Stockdale,[9] in discussing Allen's paper, argued that if the quantity of hydrogen dissolved in copper was proportional to the square root of

the gas pressure (Sieverts' law), the equation for the steam-copper reaction should read

$$2H + Cu_2O \rightleftarrows 2Cu + H_2O \qquad [1]$$

and *not*

$$H_2 + Cu_2O \rightleftarrows 2Cu + H_2O \qquad [2]$$

"If it be assumed," says Stockdale, "that under the conditions of the experiment the copper is saturated with steam, then . . . " in case 1, "the product of the oxygen content and the square of the hydrogen content should give a constant." Calculation showed that this product most certainly did not give a constant. Later work with improved apparatus, referred to by Allen in reply to the discussion of his paper, has shown this product to be constant within certain limits.

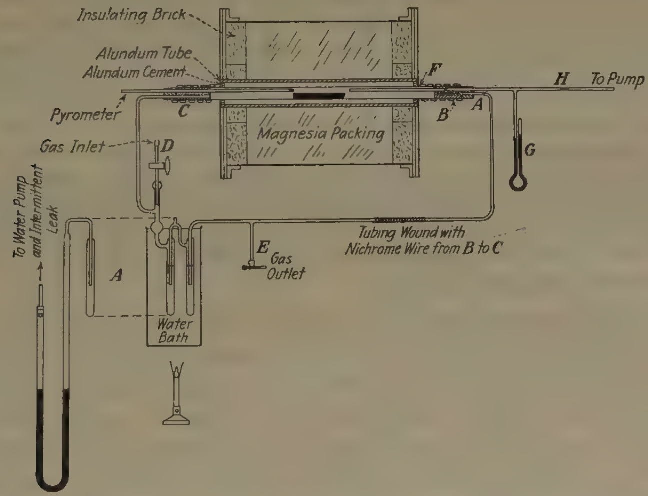

FIG. 7.—ALLEN'S APPARATUS FOR STUDY OF REACTION $2Cu + H_2O \rightleftarrows 2H + Cu_2O$

It would be expected, of course, that, given a constant amount of oxygen in the metal, the relationship

$$\frac{p_{H_2O}}{p^2_{H_2}} = K$$

would hold; whence, the steam content of an atmosphere in contact with molten copper being known, the hydrogen content of the copper could be calculated. One looks forward with considerable interest to the publication of further results by Allen on this important problem.

OVERPOLED COPPER

Just as much disagreement exists between those who have investigated the qualitative effects of gases upon copper as between those who have interested themselves in the solubilities of these gases in the metal.

Most of the former have been interested, more or less, in fire-refining and have endeavored, as a result to their experiments, to arrive at the principles underlying the methods whereby "tough-pitch" copper is obtained. The phenomenon of "overpoling" intrigued them as it does us. Because of the disagreement which still exists between different observers, it is the subject of considerable speculation. As to the appearance of overpoled copper, there is but little difference of opinion, and Percy's description, even though it first appeared seventy-odd years ago, will bear reiteration here, since it may serve to emphasize some of the troubles that are met with by those engaged in the casting of copper:

> When tough-pitch copper is kept melted under charcoal during a sufficient time and is then laded into a narrow open mould, the upper surface of the ingot when cold presents a distinct ridge along the median line. This ridge is sometimes of considerable height. In the act of solidification globules (Streu or Sprütz-kupfer of the Germans) of melted copper may be projected with force along the line on which the ridge will be found. The copper is more or less brittle, so that the ingot may be easily broken in two. Its fractured surface is more uneven than that of tough-pitch copper and appears fibrous throughout; it frequently presents small, irregular, tube-like cavities, of which the direction is from the sides and bottom of the ingot towards the median line along the upper surface, and through which the gas appears to have escaped; its color is paler and less red than the fractured surface of tough-pitch, and strongly inclines to orange. The appearance of the surface of the ingot suggests the notion that, after the upper surface had solidified in a certain degree, the still liquid, but more or less viscous, copper within had been squeezed through the upper surface along the line of least resistance.

It should be remarked that Percy was not only aware of the fact that tough-pitch copper, when melted under charcoal and poured into suitable molds, produced ingots of the type so vividly described above, but was fully alive to the equally important truth that "comparatively pure copper is not rendered brittle by being heated with comparatively pure charcoal." In other words, he appreciated the special importance of the role played by the cuprous oxide which is purposely introduced into the metal as a result of oxidation during the first period of the refining process. In fact, in the concluding paragraphs of his discussion of the phenomenon of overpoling, after dealing with the possibility of the generation of sulfurous acid as a result of interaction between oxygen and copper sulfide, he points out that, "should it hereafter be demonstrated that copper may, when melted under charcoal, take up carbon even in small quantity, carbonic oxide or carbonic acid would be formed by the action of oxygen upon the melted copper and produce the same mechanical effect as sulfurous acid;" namely, the rising of the surface and the porosity of the metal. Percy quite clearly considered sulfur dioxide to be the chief offender in this connection, though he felt that both carbon monoxide and dioxide might also be accessories to the fact. Percy's views were based in the main on the results of work conducted by Karsten(10) and Dick.(11)

Effects of Carbon Monoxide, Hydrogen, Nitrogen, etc.

At about the same time other investigators were at work, most of whom concerned themselves with the appearance and density of ingots of copper which had been melted under more or less controlled conditions. Caron[12] and Hampe,[6] for example, found that nitrogen and carbon dioxide were insoluble in copper, whereas hydrogen and sulfur dioxide were readily absorbed. They were divided in their opinion as to the behavior of carbon monoxide, the former believing it to cause the metal to spurt and rise when freezing, owing in part to the escape of carbon dioxide ($Cu_2O + CO \rightleftarrows 2Cu + CO_2$), the latter holding the view that it merely lessened the density of the metal. Dick[11] found that copper, when melted and allowed to solidify in an atmosphere of carbon monoxide, formed ingots of high density, but that if the metal were allowed to solidify in air, the density of the ingots was low.

Hampe's study of the overpoling of copper led him to the view that the characteristic appearance of ingots of such metal was due to interaction between hydrogen and cuprous oxide, which, present in different parts of the bath, were brought into contact as the metal was being poured into the ingot molds with consequent formation of steam.

Others who have studied the behavior of copper when melted in atmospheres of different gases are Lobley and Jepson,[13] Prytherch,[14] Allen[3] and Wells and Dalzell.[5] Of these, Lobley and Jepson subjected small samples of the molten metal to the action of carbon monoxide, hydrogen and nitrogen at temperatures varying from 1100° to 2300° C. for various lengths of time. The samples were allowed to freeze in these atmospheres and, when cold, were carefully examined, note being taken of their external appearance, their apparent density, their internal structure and their hardness. These workers arrived at conclusions similar to those of Caron and Hampe in respect of carbon monoxide and nitrogen; namely, that these gases are not absorbed by molten copper, up to temperatures of 1900° and 1700° C. respectively, in excess of that soluble in the solid metal. Their findings in respect of hydrogen were particularly interesting, in view of Hampe's earlier suggestions as to the important part played by this gas in overpoled copper. Lobley and Jepson found that hydrogen was absorbed by copper to a varying degree at all temperatures up to approximately 2200° C., the gas being rejected from the metal and forming blowholes while its solidification was in progress.

Prytherch's experiments were largely conducted on cathode copper, the melting of the metal being carried out in an electrically heated crucible furnace, wherein either a slightly reducing or a neutral atmosphere could be maintained. Cathode copper contains appreciable quantities of hydrogen, which result from the manufacturing process, and which are set free from the metal on melting in amounts which depend on the

conditions under which this occurs. The liberation of hydrogen from the metal during fusion under reducing or neutral conditions is by no means complete. Ingots having some of the earmarks of overpoled copper are therefore obtained.

Effect of Sulfur

Prytherch made the further experiment of bubbling sulfur dioxide through molten cathode copper, which initially must have contained appreciable quantities of hydrogen, and pouring the metal into a cast-iron mold which had previously been heated to 150° C. During solidification of the ingot a small amount of gas was liberated, but in spite of this there was no rising of the metal in the mold. Macroexamination of a section of the ingot revealed blowholes which, however, differed in shape and distribution from those caused by the passage of hydrogen through the melt.

Prytherch concluded that hydrogen and sulfur dioxide were soluble in molten copper, that the solubilities of these gases were markedly reduced at the freezing point, and that, because of this change in solubility, unsoundness resulted in ingots, even when they had been cast in iron molds. He showed, further, that the degree and the nature of the unsoundness depended on the factors, nature of gas, pouring temperature, rate of solidification, and, possibly, shape of ingot.

It is of immediate interest here to refer to the work of Skowronski(15) on the relationship between sulfur and oxygen in the melting and refining of cathode copper and on the effect of sulfur on the overpoling of copper.

In so far as the former relationship is concerned, it was shown by Skowronski that during the oxidation period of the refining operation sulfur was totally eliminated from the metal bath, being reabsorbed, however, by the molten copper when its content of cuprous oxide was low enough to allow the reversible reaction

$$6Cu + SO_2 \rightleftarrows Cu_2S + 2Cu_2O$$

to occur. In this connection Skowronski stressed the immense importance of using a low-sulfur coke or charcoal to cover the bath during the poling and casting periods of the refining process.

In so far as the effect of sulfur on the overpoling of copper was concerned, it was shown by Skowronski that sulfur is a direct agent in the overpoling of copper, the cuprous oxide left in the copper acting only as a neutralizing agent for the sulfur. He says:

> It takes a comparatively large amount of cuprous oxide to counteract a trace of sulfur; this explains why, in the refining of copper, the gases from fresh fuel will often 'overpole' the copper during the casting period, and also why copper cannot be held in the furnace at the proper pitch for more than a few hours. It also explains the limit of recharging cathode sheets into the furnace after the copper has been worked to

the proper pitch, the sulfate present in the cathode sheets overbalancing the equilibrium established between the cuprous oxide and the sulfur.

It appears that Skowronski, while admitting that the pitch of copper was "perhaps" controlled by "certain reducing gases, such as carbon monoxide and hydrogen," felt that it was sulfur *in the main* that was responsible for the overpoling phenomena with which we are here concerned.

The importance of sulfur in this connection has also been recognized by Siebe,[16] who considered sulfur dioxide to be harmful, owing to the formation of blowholes. He pointed out, however, that, if the formation of sulfur dioxide could be avoided, as, for example, by melting and casting the metal in an inert atmosphere (*e. g.*, nitrogen), the presence of sulfur was not deleterious to copper, cuprous sulfide in small quantities actually improving the mechanical properties of the metal.

Space will not allow of full comment on the above résumé of the more important observations on the effect of sulfur on the behavior of copper. One point, however, cannot be overlooked; namely, that there is almost complete unanimity as to the importance of this element. The author feels that the recent work of Wells and Dalzell[5] on the remelting and recasting in vacuo of copper containing sulfur, and the suggestions of Siebe[16] in respect of inert atmospheres are worthy of the consideration of those interested in the subject of sound copper castings. It should be noted in particular that Prytherch[14] tested out the effect of bubbling nitrogen through cathode copper which had been melted in a clay-graphite crucible and found that ingots having a density of 8.84 were obtained. Under conditions otherwise similar, but in which no nitrogen was used, the ingots had a density of 8.50.

Effect of Material of Crucible

Turning once again to Allen's work, we may note that he melted 50-gram samples of clean cathode copper in fireclay boats in contact with various gases, allowing them subsequently to cool in the gas and ultimately measuring their densities. Pure nitrogen, hydrogen, carbon monoxide and carbon dioxide were used in these experiments.

Only hydrogen gave ingots noticeably unsound, the unsoundness being so characteristic as to merit the name "hydrogen unsoundness." It differed quite markedly from the unsoundness encountered in commercial practice. The ingots which were melted in nitrogen and carbon dioxide were slightly lower in density than pure copper. Prolonged heating in carbon dioxide resulted in "a slight progressive lowering of density," according to Allen, though one would judge from his figures that the density had been little affected after the time of melting prior to freezing had been increased beyond about 10 min., the density after the 10-min. treatment being distinctly *lower* than that after longer treat-

ments. Allen attributed the lowering of density to the formation of cuprous oxide ($CO_2 + 2Cu \rightleftarrows Cu_2O + CO$), not to the presence of either carbon dioxide or carbon monoxide in the metal, ingots of copper which were melted and solidified in carbon monoxide having given the highest densities of the series (average, 8.943_3).

The work of Wells and Dalzell is of particular interest, since they took into account Caron's observation that graphite was sufficiently permeable to allow of the escape of gases from cooling and solidifying metals which had been melted in crucibles made of this material. These investigators also took particular pains in the preparation of the samples of copper which they subjected to the action of the various gases used in their experiments. On the one hand, deoxidized and desulfurized copper (density, 8.96) was employed; on the other hand, copper (0.06 per cent. O_2) which had a density of 8.95 and was produced by melting wire bar and cake drillings in vacuo was used.

Wells and Dalzell found that ingots of the deoxidized copper which been held in a state of fusion for 3 hr. in contact with any of the gases, hydrogen, carbon monoxide and carbon dioxide, the crucible being of graphite, were as dense as the untreated deoxidized copper with which they started. Nitrogen, however, caused porosity in each of the ingots tested.

When similar copper was likewise tested in vitreosil crucibles, the average densities of the ingots obtained were reduced to the amounts shown in Table 2.

TABLE 2.—*Density of Deoxidized Copper Treated in Vitreosil Crucibles*

Gas	Average Density (3 Determinations)
Carbon dioxide	6.94
Carbon monoxide	8.18
Hydrogen	8.93
Nitrogen	8.84
Steam	8.83

Wells and Dalzell argue that the absence of porosity in the ingots melted in the graphite crucible can be explained in terms of Caron's experience, and the low densities of the ingots melted in the vitreosil crucibles can be attributed to the release of the gases on the solidification of the metal. In discussing the relative effects of the various gases on the density of the samples, they say:

> Carbon dioxide has the greatest effect because of (1) the large amount of it evolved on solidification and (2) its low rate of diffusion through copper after solidification. . . . . .
>
> More carbon monoxide than carbon dioxide is released on solidification, but solid copper at 700° C. (and probably higher temperatures) is more than 28 times as permeable to carbon monoxide, hence the effect is less.
>
> Only one-sixth as much nitrogen is evolved as carbon monoxide, but the rate of diffusion of nitrogen at 700° C. is 0.0, hence practically all that is evolved is retained and therefore effective in lowering the density.

The low rate of diffusion of the nitrogen accounts, also, for the porosity of the ingots melted in the graphite crucibles.

Water diffuses through copper at a rate about 38 times that of carbon monoxide at 700° C. The amount of it evolved on solidification must be fairly large, as its effect on porosity is about the same as that of nitrogen.

The rate of diffusion of hydrogen is so great, and the amount of it evolved so small (relatively), that its effect on porosity is almost negligible.

Table 3 embodies the values on which Wells and Dalzell based their arguments.

TABLE 3.—*Used by Wells and Dalzell in Discussing Density of Copper*

Gas	Solubility at 1083° C.		Volume Released on Solidification[a]	Equiv. Volume at 1083° C.[a]	Per Cent of Volume of Copper[a]	Relative Rates of Diffusion of Gases through Copper at 700° C.[b]
	In Liquid Copper	In Solid Copper				
H_2	11.45	8	3.5	16.2	145	1000
H_2O						65
CO	24	3	21	97.2	873	17
CO_2	19.5	1.5	18	83.3	748	0.6
N_2	3.5	0.0	3.5	16.2	145	0.0

[a] K. Iwasé: Reference 2.

[b] N. B. Pilling: Action of Reducing Gases on Hot Solid Copper. *Trans.* A. I. M. E. (1918) **60**, 322.

These investigators also determined the effect of melting the oxidized copper referred to above in graphite crucibles under the four gases, carbon monoxide and dioxide, hydrogen and nitrogen. Tests were not made with steam, in view of its action on graphite. In these experiments porous ingots were obtained in three cases. The ingots which had been poured and cooled in carbon dioxide were practically equal in density to those which had been similarly produced from deoxidized copper, as also were those which had been prepared in an atmosphere of nitrogen. On the other hand, the ingots which had been poured and cooled in the reducing gases were distinctly low in density—carbon monoxide, 6.39; hydrogen, 6.60.

To explain these results Wells and Dalzell assume that "cuprous oxide is colloidally dispersed in molten copper and in that state has the property of adsorbing water and carbon dioxide." This theory, however, runs counter to nearly all the accepted opinions regarding the nature of solutions and, on this account, must be viewed with suspicion. The author feels that he, at any rate, could scarcely subscribe to the statement that "there can be little hesitancy in assuming cuprous oxide to be colloidally dispersed in molten copper." And when, after all, a much simpler

theory can be brought forward to explain these results, it seems logical to do so.

The author has noted with interest that Wells and Dalzell particularly desired to prevent appreciable reduction of cuprous oxide by the reducing gases. They say quite clearly: "Only a 5-min. exposure to hydrogen and a 2-hr. exposure to carbon monoxide were allowed in the liquid state, as appreciable reduction of the cuprous oxide was not desired." Now, had complete reduction of the cuprous oxide occurred, the ingots presumably would have been sound, since, on cooling, the gases, hydrogen and steam in the one case, and carbon monoxide and carbon dioxide[1] in the other, would ultimately have escaped from the metal, in part via the free surface, in part via the crucible walls. So long, however, as cuprous oxide was present in the melt, reactions between *dissolved* hydrogen or carbon monoxide and *dissolved* oxygen could occur. And such reactions would obviously be capable of continuing as long as liquid copper-oxygen alloy remained. In view of the fact that the solubility of oxygen in solid copper is so small, the chances in most cases would be that the water or carbon monoxide which resulted from the reactions would produce blowholes in the neighborhood of such eutectic as was present in the solid alloy. This, as Allen has also observed, is the type of unsoundness which always occurs in copper when oxygen is present. "The blowholes are smaller, more numerous, and tend to congregate in those parts of the crystal grain last to solidify, and in the crystal boundaries." Hydrogen unsoundness in chill castings is characterized by "numerous large holes having a tendency to run toward the centre of the ingot being formed," these holes being *fewer* in number, the more *slowly* the metal freezes.

A consideration of all the qualitative work described above leaves one in a state of considerable doubt as to which gases do and which gases do not enter into solution in liquid copper. If Wells and Dalzell are right in supporting Caron's views regarding graphite crucibles, the work of Lobley and Jepson must be put out of court. But doubt on the results obtained by Allen is hard to substantiate, though note may be taken of the fact that he used fireclay boats to hold his samples. In view of the fact that hydrogen caused unsoundness in the ingots he obtained, unless cooling of the metal took place slowly, it might be inferred that fireclay is relatively impervious to gases, and that the deficit in density which characterized the ingots obtained by him from the nitrogen and carbon dioxide melts was due to the partial retention of these gases in the metal on its solidification. But the density of the ingots obtained from the carbon monoxide melts approached closely to that of pure copper, whereas Wells and Dalzell's ingots (melted in vitreosil in an atmosphere of carbon

[1] Caron(12) found carbon dioxide in the gases evolved from oxidized copper which had been melted in an atmosphere of carbon monoxide.

monoxide) were exceptionally porous. It is clear that more attention needs to be paid to the effects of cooling rates on these phenomena. It may be that the many inconsistencies referred to above can be explained in terms of this important factor, which apparently was unallowed for entirely in Wells and Dalzell's experiments and appears to have been fully considered by Allen only in connection with his experiments with hydrogen.

Gases Evolved from Solidifying Copper

There still remain for consideration the results of experiments by numerous workers on the gases either evolved from copper on cooling from the liquid state or extracted from the solid or liquid metal subsequent to preparation under the usual conditions of the foundry or refining shop.

Many of the earlier workers on the subject—Lucas,[17] Erdmann and Scheerer,[18] Erdmann and Marchand[19]—thought that oxygen was the gas which ordinarily escaped from copper on its solidification; Russell and Matthiessen,[20] however, believed the usual gases to be carbon monoxide and sulfur dioxide. The author[8] found the following gases present in the mixture which *evolved* from copper that had been melted in a Schwartz furnace under usual foundry conditions: carbon monoxide, carbon dioxide, hydrogen, nitrogen, oxygen and steam. Sulfur dioxide may have been present but was measured, if present, as carbon dioxide in the analytical system used. In these experiments copper was poured from a ladle into an open sand mold 4 in. in diameter and 8 in. deep. Into the still molten copper a silica tube connected with an evacuated bulb in the manner shown in Fig. 8 was introduced. While the tube was being lowered into the copper, the stopcock *A*, through which heated air and gas could escape from the silica tube, was allowed to remain open. The tube was lowered into the metal until a volume of approximately 100 c.c. of the tube was still unfilled with metal. The stopcock *A* was closed 15 sec. after the silica tube had reached its lowest position relative to the molten copper. During this time, it was assumed that most, if not all, of the air originally in the silica tube would have been displaced by gases evolved from the metal.

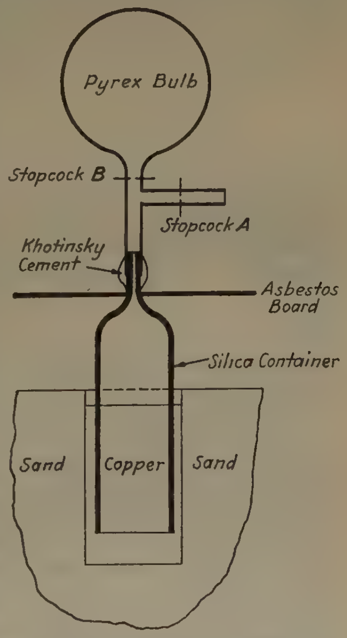

Fig. 8.—Apparatus used by Ellis in collecting gases given off during cooling of molten copper.

After a further wait of 1 min., stopcock *B* was opened and the gases above the copper distributed themselves between the container and the bulb. Stopcock *B* was allowed to remain open for 4 min. before closing. Later the bulb was separated from the container by melting the Khotinsky cement which connected them.

Table 4 shows the analyses of the mixtures *evolved* and extracted from two samples of copper (1) from a charge that consisted entirely of copper scrap and (2) from a charge which consisted in part of oxidized copper (about 0.14 per cent of oxygen).

TABLE 4.—*Analysis of Gaseous Mixtures Evolved and Extracted from Ingots of Copper during Solidification*

	Volume of Gas (Per Cent of Copper)			
	(1)		(2)	
	Evolved	Extracted	Evolved	Extracted
Carbon monoxide	1.2	9.6	0.4	2.7
Carbon dioxide	1.1	11.6	0.5	33.8
Hydrogen	1.9	nil	nil	10.1
Nitrogen	33.8	0.2	10.2	nil
Oxygen	1.6	1.9	6.1	0.2
Steam	1.6	21.0	0.9	15.5
Total	41.1	44.3	18.1	62.3

Possible sources of error in those experiments involving measurements of the gases rejected by the metal on freezing were: (1) the closing of stopcock *A* before the air which was unavoidably trapped in the silica container had been driven out; (2) the presence of charcoal within the container; (3) reaction between evolved gases within the container; and (4) the presence of gases adsorbed on the inner walls of the apparatus.

The first would lead to the inclusion of nitrogen and oxygen in the gaseous mixtures, which would otherwise have been absent. If all the nitrogen and oxygen in these mixtures had originated in air which had first been entrapped and had been held in the silica container due to premature closing of stopcock *A*, the ratio of nitrogen to oxygen would have been 79.2/20.8 = 3.81 in both cases. In mixture 1, however, this ratio is far exceeded, so much so that one cannot but believe that most, if not all, the nitrogen in the mixture was ejected during the cooling and solidification of the copper. In mixture 2 the ratio is at best less than one-half what it should be, a fact which supports the view that most, if not all, the oxygen in the mixture was dissolved in the copper and rejected therefrom on freezing.

The presence of charcoal in the container might account for the carbon monoxide and dioxide in the gaseous mixtures, and reaction between

these gases might account for the proportions in which they were found to be present. It does not appear possible that the water vapor present could be accounted for by assuming that reaction between hydrogen and oxygen had occurred within the container.

One is tempted to believe the gases quoted in Table 4 to have been largely, if not entirely, evolved from the copper subsequent to the closing of stopcock *A* (Fig. 8). While the volumes of these gases as percentages of copper are probably on the low side, since not all the gases evolved could have entered the Pyrex bulb, the relative *proportions* of the gases in the mixtures are probably correct. The differences in the compositions of the mixtures evolved are thought to be due in the main to differences in the oxide content of the copper which was under examination. It is of interest to note that the total hydrogen content of the gases evolved and extracted from the oxidized copper was definitely higher than that of the gases from the relatively deoxidized material.

Gases Extracted from Solid and Liquid Copper

A number of investigators have studied the gases *extracted* from *solid* copper. Guichard,[21] for example, heated copper in vacuo at 600° to 630° C. and extracted carbon dioxide, hydrogen and nitrogen, with a little carbon monoxide. With thick wire or rods, the gas was evolved slowly; with fine wire, more rapidly.

Of special interest are results which were obtained by Villachon and Chaudron.[22] Their experiments involved the fusion of electrolytic copper (among other metals) under reduced pressure at 1400° C. for a period of 1 hr. The pressure was $\frac{1}{50}$ mm. The metal was allowed to solidify in vacuo and was then reheated to a temperature of 600° C. and held there for some time (not stated). During this time 0.25 c.c. of hydrogen and 0.4 c.c. of carbon monoxide were removed from the metal. Villachon and Chaudron observe that the proportions of these gases removed from the solid copper at this temperature agree closely with what might be expected from a study of solubilities of these same gases in copper at the melting point (Sieverts and Iwasé). They further remark that "the only method of reducing the amounts of hydrogen and carbon monoxide contained in metal consists in heating small pieces of sheet for as long a time as possible" at temperatures "close to the melting point. Fusion in vacuo, it is true, removes nitrogen completely. Hydrogen and carbon monoxide, however, form with metals . . . dilute solutions of hydrides and carbonyls having extremely low dissociation pressures." It is to be noted that these workers do not refer to the gases extracted from the *liquid* copper during fusion at 1400° C.

Of the investigators who have studied the gases extracted from *liquid* copper under reduced pressure, only Carpenter and Elam,[23] apart from the author, appear to have attacked this problem at all thoroughly.

These investigators conducted experiments on Rio Tinto best selected copper, using the apparatus shown diagrammatically in Fig. 9, and reported on their experiments with this relatively pure metal as follows:

. . . Below are two analyses of the gases evolved from pure copper (Rio Tinto best selected), somewhat overpoled. In the first the gas was heated to constant

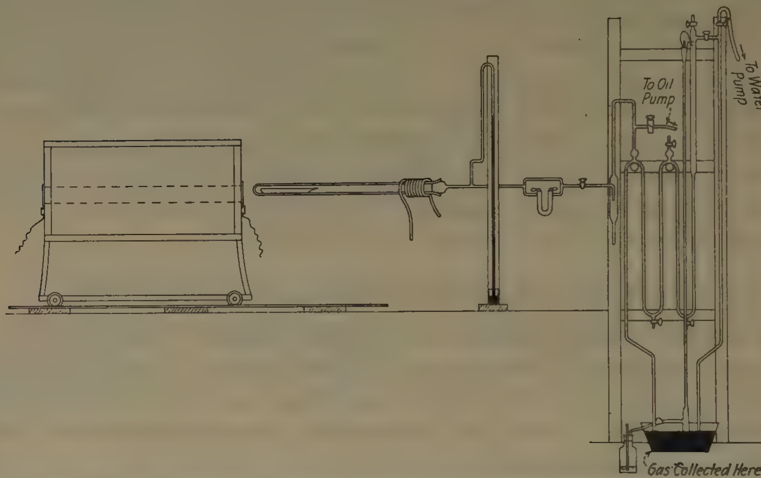

FIG. 9.—APPARATUS USED BY CARPENTER AND ELAM FOR COLLECTING GAS LIBERATED BY MELTING ADMIRALITY BRONZE. (From *Jnl.* Inst. Met. (1918) **19**, 163.)

volume. In the second it was collected as quickly as possible. They most nearly resemble the analysis of the gas from the copper-tin alloy just given, and since they are so much alike they emphasize still further the important part played by the zinc in altering the volume and composition of the gas collected from Admiralty bronze. In neither of these was there any free hydrogen.

	Gas Collected Quickly, Per Cent	Gas Collected Slowly, Per Cent
Hydrogen sulfide or sulfur dioxide	70.9	61.2
Carbon dioxide	19.6	34.9
Carbon monoxide	4.7	
Hydrogen		
Saturated hydrocarbons	0.8	
Unsaturated hydrocarbons		
Oxygen	1.2	1.1
Residual gas, nitrogen	2.8	2.8
Volume of gas collected per 100 g. metal at 0° C. and 760 mm	4.45 c.c.	5.95 c.c.

In these cases there was not much cuprous oxide in the copper, and very little oxide was found in the tube on heating.

Gases in Copper Alloys

Before dealing with the author's work on the gases evolved from copper on melting under reduced pressure, reference may well be made to the work of Guillemin and Delachanal[24] and of Guillet and Roux,[25] and to further experiments by Carpenter and Elam on alloys of copper containing tin and zinc.

Guillemin and Delachanal examined certain bronze forgings and found them to contain hydrogen, carbon monoxide and carbon dioxide. They noted that the ratio of the hydrogen to the carbonic gases was considerably greater in the sound than in the unsound forgings, the hydrogen content of the gaseous mixture extracted from the sound forgings being 90 per cent.

Carpenter and Elam found the volume and composition of the gases extracted from molten Admiralty bronze (88:10:2) to vary with the method of collection, *depending on the time the gas remained in the furnace.* Their apparatus is shown in Fig. 9. When the gas was removed quickly, the total volume collected was small and the mixture contained very little hydrogen and generally a fair proportion of sulfur dioxide and hydrogen sulfide. When the gas was heated to constant volume or removed only slowly, the volume collected was much larger and the mixture contained about 50 per cent of hydrogen. In this case, sulfur dioxide and hydrogen sulfide were either very low or entirely absent. Carpenter and Elam considered that zinc was largely responsible for the differences which they observed, since they found much less difference in the gases collected from copper (*vide supra*) in both ways. Zinc, they believed, might act (1) by lowering the solubility of the gases in the alloy and (2) by volatilizing and reacting with the sulfur evolved to form zinc sulfide.

Guillet and Roux concerned themselves, among other things, with the composition of the gases extracted from solid brasses containing 60 and 67 per cent of copper, respectively, at a temperature of 720° C. for 2 hr. The volumes of gas given off were 39 per cent and 55 per cent of the volumes of each of the alloys examined. In the 60 per cent alloy, the gaseous mixture contained 8.6 per cent CO_2, 56.4 per cent H_2 and 43.6 per cent N_2 (by difference); in the 67 per cent alloy, the mixture comprised 13.6 per cent CO, 14.16 per cent CO_2, 65.4 per cent H_2 and 7.4 per cent N_2.

On heating brass (composition not stated) to 1040° C. at a rate of 7° C. per minute, evolution of gas was first found to occur at 530° C. and then to cease at 1040° C. The volume of gas extracted was one-half the volume of the metal. The gaseous mixture contained 9.5 per cent CO, 19.0 per cent CO_2, 35.9 per cent H_2, 22.2 per cent CH_4 and 13.4 per cent N_2.

The extent to which the investigations on gases extracted from alloys of copper can be used in arguments regarding the validity of the analyses of gaseous mixtures extracted from more or less pure copper is limited indeed, as the experience of workers who have examined both copper and copper alloys in this connection has shown (unpublished work carried out at the Research Laboratories of the Westinghouse Electric and Manufacturing Co.). See also remarks by Carpenter and Elam quoted on p. 20. This experience finds considerable support in Guillet and Roux's results; the quite definite differences between the analyses of the gaseous mixtures extracted from the two brasses investigated by them under practically identical conditions is quite in line with this experience. While admitting the possible unreliability of the results reported to this Institute by the author(8) in 1928 (the causes of this possible unreliability are dealt with in detail below), the following remarks of Allen(3) in this connection are somewhat misleading, particularly when considered in the light of the detailed account given above of the work on the subject now in question:

> The results of analyses of gases withdrawn from solid samples of copper by heating under reduced pressures, which is the third line of investigation, afford no decisive judgment on the question. Such work has been carried out by Guichard, Carpenter and Elam, Ryder,[2] and (on brasses and bronzes) by Guillemin and Delachanal. In every case hydrogen, carbon monoxide, carbon dioxide, and nitrogen were present in the gases withdrawn, hydrogen predominating as a rule, but the quantities and compositions of the gaseous mixtures obtained varied so erratically as to inspire little confidence, and there can be little doubt that, in view of the enormous difficulty of this class of work, experimental error accounted for a large proportion of the variations. Frequently the quantities of gas obtained were much larger than would be anticipated from the solubility measurements of Sieverts and Iwasé.
>
> The recent improvements in vacuum and melting technique hold out great promise that the difficulties will be eventually overcome, but the first results are disappointing. The work of Guillet and Roux confirms that of the earlier investigators in that the gas obtained was mainly hydrogen, whilst the quantity was more in agreement with the solubility measurements of Sieverts and Iwasé, but that of Ellis is difficult to understand, for the gas he succeeded in extracting from many samples of molten copper consisted mainly of carbon dioxide, carbon monoxide, water vapour and nitrogen, with not inappreciable proportions of free oxygen. A recent paper by Villachon and Chaudron reports results similar to those of Guillet and Roux.

Gases in Casting Copper and Overpoled, Fire-refined Copper

In conclusion reference may be made to the work on gases in casting copper and overpoled fire-refined copper, which was described by the author(26) in 1928 and 1929. In the first place, a series of small ingots of copper, which had been melted in an electric furnace of the indirect-arc type under *severe* oxidizing conditions, was examined. The conditions

[2] Ryder made an analysis of the gases extracted from liquid copper but published no results in the paper referred to by Allen.

were exceptional in that air was blown through the copper for 20 min. prior to its being tapped from the furnace. One sample was poured direct from the furnace into a small graphite mold in such a way that little or no contact between the metal and the mold could occur, owing to the large amount of slag which flowed from the furnace into the mold with the metal. The sample was found to be completely enveloped in slag and to show no evidence of unsoundness. The analysis of the gaseous mixture extracted from this ingot as percentage of copper was as follows: O_2, 57.5 per cent; CO, 26.9; CO_2, 35.4; H_2, nil; N_2, 54.0; Water vapor, 12.5.

This analysis may be taken as representative of the analysis of the gaseous mixtures extracted from all the ingots made in this particular series of experiments. These mixtures contained little or no hydrogen, relatively large amounts of oxygen, carbon monoxide, carbon dioxide and nitrogen, and an average amount of approximately 112 volumes of water vapor in per cent of copper.

This brings up the question of the reliability of the results recorded. No one, the author believes, has doubted that the gases referred to were analyzed as such, though objection can be made to any system which fails to measure the quantity of sulfur dioxide extracted from copper. This gas, it may be noted, would have been measured, if present, together with the carbon dioxide, so that the figures quoted for carbon dioxide (as pointed out by the author when introducing TECH. PUB. 123 at the 1928 meeting in Philadelphia) may have represented carbon dioxide plus sulfur dioxide.

The main criticisms which have come to the author's attention, however, have been directed (1) against the use of refractory oxide crucibles, which places the dependence for the removal of oxygen from the metal entirely on the partial pressure of this gas,[3] and (2) against the inference that the composition of the gases removed from a metal by heating in a vacuum represent the mode of combination of the elements in the solid metal used as a sample.

Dealing with the first of these criticisms, it has been suggested that the dissociation pressure of cuprous oxide at temperatures corresponding to those of vacuum fusion analysis is very low as compared with the pressures that can be realized in the apparatus which, for example, was used in the extraction of the gases from the oxidized copper now in question. These pressures were of the order of 5×10^{-4} mm. of mercury.

The dissociation pressure of cuprous oxide has been calculated by a number of observers. Stahl,[27] for example, has given the values, 3.3 $\times$

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[3] It should be noted that the last thing the author desired in his experiment was to reduce the quantity of cuprous oxide present in his alloys. It was his hope that the alloy of oxygen and copper would contain after vacuum fusion the same amount of cuprous oxide as it did before.

10^{-7}, 3.6×10^{-6}, 2.8×10^{-5}, and 1.8×10^{-4} atmospheres for the dissociation pressure of cuprous oxide at the temperatures 1300° Abs. (1027° C.), 1400° Abs. (1127° C.), 1500° Abs. (1227° C.) and 1600° Abs. (1327° C.) respectively. These correspond to the following pressures in milligrams mercury: 2.51×10^{-5}, 2.74×10^{-4}, 2.13×10^{-3}, and 1.37×10^{-2}. Stockdale gives the values, 7.6×10^{-5} mm. (1200° Abs., 927° C.), 4.0×10^{-3} mm. (1350° Abs., 1077° C.), 1.9×10^{-2} mm. (1420° Abs., 1147° C.), and 1.0×10^{-1} mm. (1500° Abs., 1227° C.), for the dissociation pressures of cuprous oxide at various temperatures. Dean[28] gives the value, 5.6×10^{-3} atm., or 4.3 mm., for pO_2 ($2Cu_2O = 4Cu + O_2$) at the melting point of copper. On working through Dean's calculations, however, it was found that an appreciable error had been made, so that Dean's value must be considered as incorrect. Randall, Nielsen and West[29] quote the following equation:

$$\Delta F_T = -41{,}166 - 1.27T\ln T + 0.00037T^2 - 0.00000018T^3 + 27.881T$$

for the reaction, $2Cu + \frac{1}{2}O_2 = Cu_2O$, the copper being in the solid state. This equation gives a value for ΔF_{1356} of $-15{,}468$, whence can be obtained the value, 9.29×10^{-4} atm. for the oxygen pressure (pO_2) of pure Cu_2O at the melting point of copper. This is equivalent to a pressure of 7.06×10^{-1} mm. The author, using the data collected by Randall, Nielsen and West, has obtained a slightly different value for ΔF_T, with the result that he has obtained a value for pO_2 at the melting point of copper of 9.19×10^{-3} mm., which is in fairly close agreement with the value obtained by Stockdale.

But cuprous oxide is soluble in molten copper and, as a result, the readiness with which it dissociates is reduced. Thus, the value for pO_2 at the melting point of copper, the copper being in the liquid state and the amount of cuprous oxide in solution being small, would be of the order 1×10^{-6} mm. When larger quantities of cuprous oxide were dissolved, pO_2 might approach in value the pressures used in vacuum fusion, but the chances of deoxidizing copper by merely subjecting it to vacua of the order of 5×10^{-4} mm. seem rather remote, though not altogether impossible.

The possibility is certainly not so remote in the present state of our knowledge as to enable one to say definitely that the oxygen which was extracted from the highly oxidized copper now under discussion did not result from the dissociation of dissolved cuprous oxide. The fact that the volume of oxygen extracted from copper melted under highly oxidizing conditions (*e. g.*, ingots 1 to 3—83.4 oxygen in percentage volume of copper) was much higher than that extracted from copper melted under less severe conditions (*e. g.*, ingot 7—7.7 oxygen in percentage volume of copper) lends some support to the view that dissociation occurred. Other evidence could be quoted which also lends favor to this view. Attention should be given, however, to the fact that some oxygen was

found in the gases *evolved* from copper (see Table 4), though it must be admitted that the conditions under which the gaseous mixture was collected in this case did not entirely exclude the possibility of oxygen having been left in the container.

The situation is such that one is inclined to suggest as a problem of prime importance the investigation *per se* of the deoxidation of copper in vacua of the order of 5×10^{-4} mm. It may be noted that the rate at which deoxidation in vacuo can be accomplished must certainly be slow, in view of the fact that most of the 1½ to 2-lb. samples of copper dealt with by the author were in a state of fusion and agitation under the influence of the induced current for at least two hours.

Turning now to the inference[4] that the composition of the gases removed from the metal by melting in vacuo represents the mode of combination of the elements in the solid metal used as sample. Criticism of this inference is certainly justified in view of the methods which had of necessity to be adopted in obtaining the various samples of copper investigated by the author in the experiments described in the paper on Absorbability of Gases in Casting Copper.[(26)] The same remarks do not apply to the paper on Gases in a Sample of Overpoled Fire-refined Copper.

In all the experiments referred to in the paper first mentioned, copper was poured from a furnace (electric or oil) into receptacles wherein it was brought into intimate contact with carbon, the practice in the foundry being followed as closely as possible. This practice involved the conveyance of the copper from the furnace in ladles, the surface of the copper being covered with an ample supply of charcoal. In pouring the small ingots which were the subject of investigation, the possibility that small particles of charcoal might have been entrapped in the copper must be admitted, though it should be noted that the ingots were ground deeply before they were melted in vacuo. The object of this grinding was to remove all sand and nonmetallic materials as adhered to their surfaces. To the author's knowledge, therefore, no charcoal entered the vacuum furnace, nor did macroscopic or microscopic examination of sections of the ingots show evidence of free carbon. The possibility that some charcoal did enter the furnace cannot, however, be entirely disproved.

Per contra, it is of interest to see, as mentioned in the same paper, that when such samples of oxidized copper as had been conveyed under a charcoal cover from the furnace to the molds in such a way as to insure intimate contact between the carbon in the cover and the cuprous oxide in the melt were remelted and degasified, the gases extracted always contained about the same weight of combined carbon as percentage of

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[4] It may be pointed out that the author had never intended to suggest that the gases analyzed existed entirely in the metal sample in that form. A reading of his paper on Absorbability of Gases in Casting Copper,[26] however, gives the impression that such was his view.

copper. It seems unlikely that this relatively uniform content of carbon would have been found in the gases, had charcoal been entrapped in the ingots and given opportunity to react with cuprous oxide in the melt during the vacuum fusion. One is tempted to argue that, while the proportions of carbon monoxide and carbon dioxide extracted from the ingots may have been determined by conditions over which there was no control in the vacuum fusion apparatus, the amount of carbon extracted from the copper in combination with oxygen as carbon monoxide and carbon dioxide might have been determined, for example, by the time during which the oxidized copper was in contact with charcoal in the ladle. Since this time varied but slightly in these experiments (from 4 to 8 min.), the carbon content might be expected to vary but slightly, too. It is difficult, however, to give a satisfactory explanation of this uniformity in carbon content, but *the fact that the carbon content was so uniform completely upsets any suggestion that the carbonic gases extracted from the copper resulted from reaction between entrapped charcoal and cuprous oxide during fusion in the vacuum apparatus.*[5]

In so far as water vapor is concerned, the question arises as to how far the water vapor *extracted* (using this term rather loosely) from the oxidized copper samples resulted from reaction between hydrogen and cuprous oxide in the vacuum furnace. The fact that hydrogen was absent from the gases extracted from the most highly oxidized samples would tend to support the view that the water vapor present in these same gases resulted largely from reaction between hydrogen and cuprous oxide. On the other hand, it is of interest to note that the samples of copper melted in the electric furnace *under reducing conditions* contained neither hydrogen nor carbon monoxide but *did contain* oxygen, carbon dioxide, nitrogen and water vapor. Note may also be made of the fact that the gases *extracted* from the ingot of overpoled copper referred to in the paper on Gases in Overpoled Fire-refined Copper[26] comprised carbon dioxide and water vapor. These facts certainly lend support to the hypothesis that reactions between cuprous oxide (or oxygen) and carbon monoxide and hydrogen do take place during fusion in the vacuum furnace with the formation of carbon dioxide and water vapor. It seems unlikely, however, that the carbon monoxide found in many of the gaseous mixtures can be accounted for by any reaction within the melt.

Conclusion

In conclusion, it will have been made abundantly clear that much has yet to be done ere the problem of the composition of the gases dissolved in copper is completely solved. The widely opposed views regarding the

[5] The author assumes in this argument that the carbon dioxide measured in the analytical system of the apparatus was uncontaminated by sulfur dioxide: *i.e.*, all the sulfur present was combined with copper as Cu_2S ($6Cu + SO_2 \rightleftarrows 2Cu_2O + Cu_2S$).

solubility, for example, of carbon monoxide in copper (Slavinski, Gorshkow and Syromiatnikow,[30] for instance say quite definitely, "It has been proved experimentally that CO is insoluble in copper."), the divergence of opinion as to the behavior of nitrogen and the differences in belief regarding the role of sulfur make it quite evident that further facts are required. Those who have concerned themselves with the phenomenon of overpoling and have shown beyond a peradventure the important role played by hydrogen in this connection have, in the author's opinion, been tempted to view as heresy all statements to the effect that gases other than hydrogen can have any effect on the porosity of copper. Too many experiments, however, have been conducted, in which other gases than water vapor have been found in the mixtures evolved or extracted from copper, for these to be lightly set aside. The fire-refining of copper is an important process, but hundreds of tons of castings are being made in such furnaces and under such conditions as have been rather roughly described in the author's paper on Absorbability of Gases in Casting Copper. These conditions differ materially from those existing in fire-refining practice, and while the author freely admits that the presence of hydrogen in copper is the most likely cause of the troubles met with in the casting of copper under more or less ordinary foundry conditions, he does not feel that sufficient evidence has even yet been produced to enable one to say that such gases as carbon monoxide, nitrogen and sulfur dioxide are quite innocuous.

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24. Guillemin and Delachanal: *Compt. rend.* (1908) **147**, 1309; (1910) **151**, 881.
25. Guillet and Roux: *Compt. rend.* (1927) **184**, 724.
26. Ellis: *Trans.* A. I. M. E., Inst. Met. Div. (1929) 443, 470.
27. Stahl: *Metallurgie* (1907) **4**, 682.
28. Dean: U. S. Bur. Mines *Inf. Circ.* 6395 (1930).
29. Randall, Nielsen and West: *Ind. & Eng. Chem.* (1931) **23**, 388.
30. Slavinski, Gorshkow and Syromiatnikow: *Jnl.* Inst. Met. (1930) **43**, 155.

DISCUSSION

(*E. C. Bain presiding*)

C. R. Hayward, Cambridge, Mass. (written discussion).—The author has performed a distinct service to copper metallurgists by bringing together the results obtained by various investigators of this important subject. The apparently conflicting conclusions obtained, as Mr. Ellis points out, are frequently due to the widely differing conditions under which the various experiments were performed. Some of the investigators have tried to apply their findings to copper refining in an attempt to explain certain well-known phenomena, but in so doing have overlooked an important modifying influence.

In discussing a paper by Pilling and Halliwell in 1926 (*Trans.* A.I.M.E., **73**, 695) I made the following statement: "We also forget that there is often a change in the composition of copper as it flows from the furnace to the mold and during setting. Most of our practical copper metallurgists minimize the absorption of oxygen during this period but I have made tests which show that the absorption is considerable and I feel confident that it has a bad effect upon the final properties of the copper as the fabricator has to receive it." I still hold that opinion.

In order to explain the phenomena accompanying the casting of overpoled copper, we must consider: (1) the gases present in the copper when it leaves the furnace, (2) the action of the atmosphere on the copper as it flows from the furnace, and (3) the gases formed by reactions after the copper leaves the furnace. We can completely explain the porosity of cast overpoled copper if we admit the importance of oxidation during casting, and out of the conflicting data received by Mr. Ellis admit the following statements to be true:

1. Hydrogen and carbon monoxide are readily soluble in molten copper and have a rapid diffusion rate.

2. Sulfur dioxide, water vapor, carbon dioxide and nitrogen are less readily soluble in molten copper than hydrogen and carbon monoxide and their diffusion rate is less.

At the conclusion of the poling period, it will therefore be probable that the gases contained in the copper are principally carbon monoxide and hydrogen. Sulfur is also present. As the copper flows from the furnace there is a rapid absorption of oxygen which either directly, or through intermediate copper oxides, reacts with the sulfur, carbon monoxide and hydrogen, producing sulfur dioxide, carbon dioxide and water vapor. All of these being only slightly soluble in the cooling copper and having slow diffusion rates, create gas pockets and produce the well-known appearance of overpoled copper. The hydrogen and carbon monoxide that are unaffected will to a considerable extent under the relatively slow cooling of commercial practice, diffuse without leaving serious porosity.

My belief in this theory is confirmed by the fact that if copper is cast in a reducing atmosphere into a warm mold and air is rigidly excluded during solidification, only slight porosity is obtained.

N. A. Ziegler, East Pittsburgh, Pa. (written discussion).—A person who never has done any work in this line may wonder why, after investing so much effort and money on this problem, different investigators with well-known names get results differing so widely. The present writer feels that a factor that would explain at least part of the difficulties is the purity of original material. Yensen was first, perhaps, to point out specifically that we know practically nothing about pure metals and that most likely their properties are entirely different from what they are commonly supposed to be.[6] Yensen is working in ferrous metallurgy and all his publications refer specifically to iron and its alloys, but principles and ideas suggested by him are applicable to any metal.[7]

Whether or not Yensen is correct in his theories remains to be seen, but the main fact, demonstrated by him a number of times, and at present beyond any doubt, is that even traces of foreign elements (less than 0.01 per cent in magnitude) have sometimes a tremendous effect on properties of metals.

The writer has demonstrated sufficiently convincingly, it is hoped, that solubility of oxygen in iron, with less than 0.01 per cent of other interstitial impurities, at temperatures over 900° C., amounts to about 0.1 per cent. But if the amount of these impurities increases, this solubility figure decreases rapidly.[8] Would it not be logical to expect something similar in case of copper? At present we are not prepared to give a definite answer, because copper cannot be analyzed with sufficient accuracy; anything that contains less than 0.005 per cent foreign elements would be considered as "pure copper," and yet it is not so.

For example, carbon is considered as being insoluble in copper and not having any effect on its properties. It may be correct in most practical cases, but, on the other hand, it must be admitted that there is no absolute insolubility of one element in the other. Hence, copper-carbon constitutional diagram must exist, although the amounts of carbon may be expressed, perhaps, in tenth-thousandths of a per cent.

The same may be true about many other elements. If this viewpoint were accepted, instead of "pure copper" we should deal with rather complicated alloys. The relative amounts of foreign elements present in them may have a wide variation, but the total sum remains about the same, which would, perhaps, make some experimenters believe that samples of "copper" with which they are dealing are uniform in composition.

In work on gases in metals, there are many interfering factors, some of which are yet beyond our control, and which, owing to their small values, are very difficult to conquer. This viewpoint may at least partly explain disagreement in the results of some well-known experimenters summarized with such remarkable skill in Mr. Ellis' paper.

R. C. Dalzell, Trenton, N. J. (written discussion).—Mr. Ellis has suggested (p. 500) that the discrepancy between our results[9] and those of Allen are due to

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[6] T. D. Yensen: What is Magnetic Permeability of Iron? *Jnl.* Franklin Inst. (1928) **206**.

[7] T. D. Yensen: Pure Iron and Allotropic Transformations. *Trans.* (1929) **84**, 320; On the Road to Pure Iron or Some of Its Indicated Properties. *Trans.* Amer. Electrochem. Soc. (1929) **56**, 215.

[8] N. A. Ziegler: Solubility of Oxygen in Solid Iron. *Advance Copy* Amer. Soc. Steel Treat. (1931).

[9] Wells and Dalzell: *Trans.* A. I. M. E. (1930) **89**, 349.

different cooling rates. I believe that this is correct. Our own cooling rates were perfectly uniform.

On page 508 Mr. Ellis states that Cu_2O is soluble in molten copper. All present data seem to indicate this. As pointed out in our paper, however, it seems possible that colloidally dispersed Cu_2O might give the same effects as a true solution would give. The theoretical objections raised on this point by Mr. Maier and others have not convinced me that it is impossible. Perhaps I should feel that unless I can prove this basic assumption which seems so radical I should abandon the theory I have proposed. The ability to explain all the phenomena of fire-refining and casting of refined copper, however, encourages me to hold to my ideas until absolute negative proofs or a better theory are advanced. Any acceptable theory must explain, not several, but all of the following:

1. Why proper pitch may be obtained by poling.
2. What oxygen content has to do with pitch.
3. What overpoling actually is.
4. Why slight reoxidation will not perfectly rectify the overpoled condition.
5. Why mold temperature and pouring temperature affect the pitch.
6. Why deoxidized copper forms dense castings.

As to the subject in general, I am primarily interested in refined copper. Even if my basic interest were a copper alloy or an impure copper, I should surely start a prolonged investigation with the purest metal obtainable. After this had been studied thoroughly it would be time to test the effect of various impurities or alloying elements.

I admire Mr. Ellis' apparatus for collecting the gases evolved by solidifying metal. I should like to see it used for studying copper throughout the cycle of melting, refining, and casting refined copper, preferably including overpoling. I object to small-scale work on this phase because there are too many variables that might be overlooked. I believe, though, that other aspects of the problem can be worked out in a laboratory-size Ajax-Northrup high-frequency furnace.

O. W. Ellis (written discussion).—The discussions do not in any way conflict with statements made in the paper, if exception be made of Mr. Dalzell's further support of his theory. It appears to me that Dr. Hayward's theory explains quite as completely as Mr. Dalzell's the various points (1 to 6) raised by Mr. Dalzell in his discussion. There is no doubt whatever that Mr. Ziegler is right in saying that the attainment of even greater degrees of purity in certain of the nonferrous metals will result in the discovery of most interesting phenomena.

Chemical and Physical Features of Copper Leaching

By John D. Sullivan,* Columbus, Ohio

Leaching of copper ores is a comparatively old art, probably dating back to medieval times. The leaching of mixed oxidized-sulfide ores, however, is modern. The first modern plant leaching mixed ores was at Ajo, Ariz., where commercial operations commenced in 1917. While the ore at Ajo was primarily oxidized there was always some sulfide present. The Inspiration leaching plant, which was put in operation in 1926, may be considered the first plant constructed for the extraction of both sulfide and oxidized copper. The pioneers in leaching mixed ores had a difficult task before them since they had very little fundamental data on the chemical and physical factors of leaching. For the very large amount of painstaking experimental work that was done before building the Ajo and Inspiration leaching plants the reader is referred to the original articles.[1]

For a period of several years the writer worked on problems connected with the chemistry and physics of leaching mixed oxidized-sulfide ores. This paper summarizes the experimental work and discusses various factors involved in leaching ores. It also presents many data heretofore unpublished. Among the new data are those on chrysocolla, pages 518 to 519, inclusive; dioptase, page 519; covellite, page 529; virtually all those on chalcopyrite, pages 531 to 533, inclusive; enargite, page 533; and tetrahedrite and tennantite, pages 534 and 533.

The oxidized copper minerals of commercial importance in American ores are azurite, $Cu_3(OH)_2(CO_3)_2$; malachite, $Cu_2(OH)_2CO_3$; chrysocolla, $CuSiO_3.2H_2O$; tenorite (or melaconite), CuO; and cuprite, Cu_2O. Dioptase, $CuSiO_3.H_2O$ is less common. Brochantite, $CuSO_4.3Cu(OH)_2$, while the chief mineral in the ore at Chuquicamata, is not of major importance in the United States. Metallic copper, while chemically not so, is frequently classed as an oxidized mineral of copper. The most important sulfide minerals are chalcocite, Cu_2S; covellite, CuS; bornite, Cu_5FeS_4; and chalcopyrite, $CuFeS_2$. Less important complex sulfides are enargite,

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* Battelle Memorial Institute.

[1] H. W. Morse and H. A. Tobelmann: Leaching Tests at New Cornelia. *Trans.* A.I.M.E. (1916) **55**, 830.

H. A. Tobelmann and J. A. Potter: First Year of Leaching by the New Cornelia Copper Co. *Trans.* A.I.M.E. (1919) **60**, 22.

G. D. Van Arsdale: Leaching Tests at Inspiration Cons. *Arizona Mining Jnl.* (March 30, 1926).

Cu_3AsS_4; tennantite, $Cu_8As_2S_7$; tetrahedrite, $Cu_8Sb_2S_7$; chalmersite, $CuFe_2S_3$; and cubanite, $CuFe_2S_4$.

In general it may be said that the best commercial solvent for oxidized minerals is sulfuric acid, and for sulfide minerals acidified ferric sulfate. Ferrous sulfate, however, is not a solvent for sulfide copper minerals. Other solvents may be chemically equivalent to those mentioned but for ease of preparation and handling, and cheapness, none excel. Since many ores contain sulfide minerals, especially pyrite, solvent may be generated in the leaching process. This is especially true in heap-leaching operations. For native copper and some carbonate ores an ammoniacal solution of cupric ammonium carbonate is frequently used. A slimy ore containing many fines is usually unsatisfactory for ammonia leaching, owing to the high loss of leaching solvent in the ore by adsorption. A slimy ore was leached by the ammonia process at Bwana M'Kubwa,[2] but the copper content of the ore was high, so that this ore could economically stand a larger solvent loss than low-grade ores.

Chemistry of Dissolution of Copper Minerals

Fundamentally, leaching is a chemical process and even the type of leaching selected for a given ore may be governed largely by the chemistry of dissolution of the copper minerals in the ore. A basic study of the chemistry of leaching copper ores can best be made by a study on copper minerals, preferably as pure as possible. The rate of dissolution of a given mineral in an ore may, of course, be somewhat influenced by association of gangue or other minerals. A study should consist not only of the rate of dissolution of given minerals in various reagents at different temperatures, but it should also consider the mechanism of dissolution.

In this paper data are given on the dissolution of some copper minerals. Table 1 gives chemical analyses for the various minerals discussed. They are reproduced here so that the reader may judge the purity of the materials investigated. The analyses are given in most instances for the minus 100 plus 200-mesh size. The chemical analysis varied very little with size because large pieces of minerals were crushed and screened. The data presented in this paper on copper minerals were obtained in experiments made in a room with temperature thermostatically controlled. Unless stated to the contrary, all temperatures were 35° C., the average in the Southwest. Footnote references are given to former articles that describe experimental technique and procedure in detail.

The rate of dissolution of minerals may be influenced by the rate of agitation. Samples and solutions were placed in 5-pint (acid) bottles that rotated on revolving rolls at the rate of 5¼ r.p.m. The bottles

[2] C. S. Van der Poel: Metallurgical Operations at Bwana M'Kubwa. Paper presented at Third Empire Mining and Metallurgical Congress, South Africa, 1930.

TABLE 1.—*Chemical Analyses of Minerals*

Mineral	Analysis, Per Cent							
	Cu	Fe	S	Insol.	Mn	CaO	MgO	Other Constituents
Azurite	55.35	0.23	0.02	0.46	0.17	0.32		
Malachite	55.43	0.49	0.05	0.42	0.24	0.56		
Tenorite	29.86	4.35	tr	26.88	5.0			
Chrysocolla	33.52	0.17				1.11	0.61	
Cuprite	77.08	0.62	tr	10.68				
Chalcocite	78.28	0.21	20.95	0.54				
Bornite, Magma	65.20	10.15	24.18	0.48				
Bornite, Sunset	60.54	12.00	25.54	0.96				
Covellite, Kennecott	66.39	0.45	31.94	0.34				
Covellite, Butte	64.46	0.98	31.72	2.76				
Chalcopyrite	31.34	31.00	34.06	2.32				Zn 1.20
Enargite No. 1	45.02	1.10	31.80	2.32				As 13.22, Sb 0.69
Enargite No. 2	45.12	0.08	31.36	2.88				As 17.37; Sb 0.56
Tennantite	41.51	0.30	25.30	0.76				As 15.35; Sb 1.02; Pb 1.80

were not stoppered, thus permitting free access of air. Experiments made under different conditions of agitation will be referred to in the discussion of various minerals.

Azurite and Malachite.—Azurite and malachite sized to minus 100 plus 200 mesh were completely dissolved in one hour in solutions containing 1 to 5 per cent of sulfuric acid. The reaction for the dissolution of azurite may be written:

$$Cu_3(OH)_2(CO_3)_2 + 3H_2SO_4 = 3CuSO_4 + 2CO_2 + 4H_2O$$

For malachite:

$$Cu_2(OH)_2CO_3 + 2H_2SO = 2CuSO_4 + CO_2 + 3H_2O$$

The theoretical consumption of acid is 1.54 parts of H_2SO_4 per part copper dissolved. For azurite of the composition given in Table 1, the consumption of acid was approximately 1.6 parts of H_2SO_4 per part of copper extracted. For malachite, the consumption varied from 1.62 parts for 1 per cent acid to 1.71 for 5 per cent acid.[3]

Tenorite.—The reaction for the dissolution of tenorite (melaconite) may be written:

$$CuO + H_2SO_4 = CuSO_4 + H_2O$$

The mineral shown in Table 1 was not of high purity. About 98 per cent of the copper in minus 100 plus 200-mesh material was rendered

[3] J. D. Sullivan: Dissolution of Various Oxidized Copper Minerals. U. S. Bur. Mines *Rept. of Investigations* 2934 (1929).

soluble in one hour by a solution containing 1 per cent of sulfuric acid.

Chrysocolla.—Data are given in Fig. 1 showing the rate of dissolution of pieces of mineral of various sizes in 1 per cent sulfuric acid solution. The rate of dissolution was even more rapid in solutions containing 2 and 5 per cent H_2SO_4. Minus 0.525-in. plus 3-mesh size was 51 per cent dissolved in 6 hr. and completely in solution in 30 hr. in 5 per cent H_2SO_4. Minus 10 plus 28-mesh pieces were 97 per cent dissolved in 6 hr. in 5 per cent acid. Sizes smaller than 28 mesh were completely dissolved in 6 hr. or less in either 2 or 5 per cent H_2SO_4.

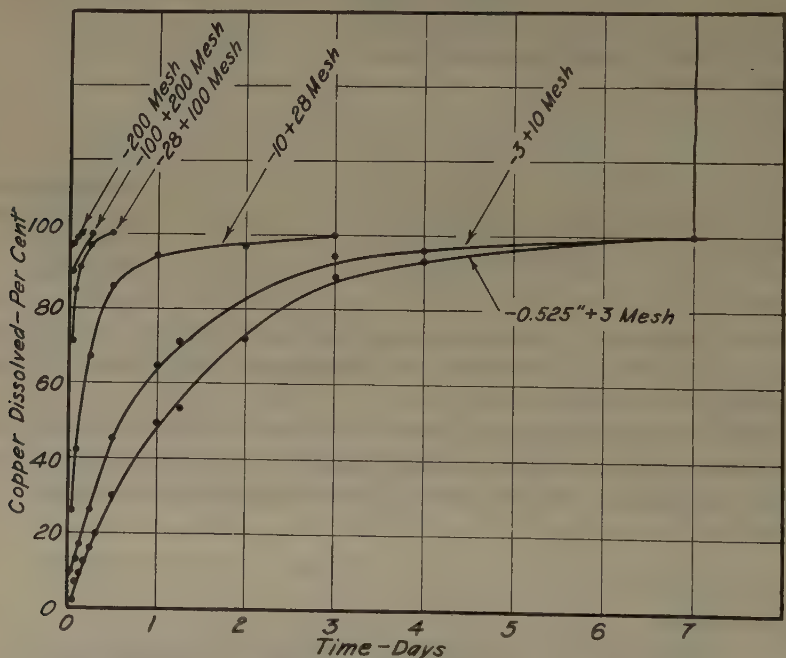

FIG. 1.—DISSOLUTION OF VARIOUS SIZES OF CHRYSOCOLLA IN 1 PER CENT SULFURIC ACID.

While the rate of dissolution increased with decreases in size of particles, the difference was not so great as one might expect, since a large increase in surface area per unit weight is normally produced by crushing and grinding. However, owing to internal fractures and porosity, surface areas cannot be predicted without actual measurements. Such measurements have not been made for chrysocolla.

The consumption of acid in dissolving minus 100 plus 200-mesh material was somewhat higher than theoretical, and increased with increases in strength of acid. Presumably some silica was dissolved to form silicic acid. The lime and magnesia in the mineral also consumed acid. The ratio, by weight, of acid consumed to copper extracted was

1.77, 1.90 and 2.06 for 1, 2 and 5 per cent acid, respectively. The reaction for the dissolution of chrysocolla in sulfuric acid can probably be written:

$$CuSiO_3.2H_2O + H_2SO_4 = CuSO_4 + SiO_2 + 3H_2O$$

Dioptase.—Data are given in Fig. 2 on the rate of dissolution of dioptase in 1, 2 and 5 per cent sulfuric acid. Dioptase is dissolved much more slowly than chrysocolla. The term "chrysocolla" is frequently given to all silicates of copper. The remark is often made that some forms of chrysocolla are soluble in sulfuric acid whereas other forms are

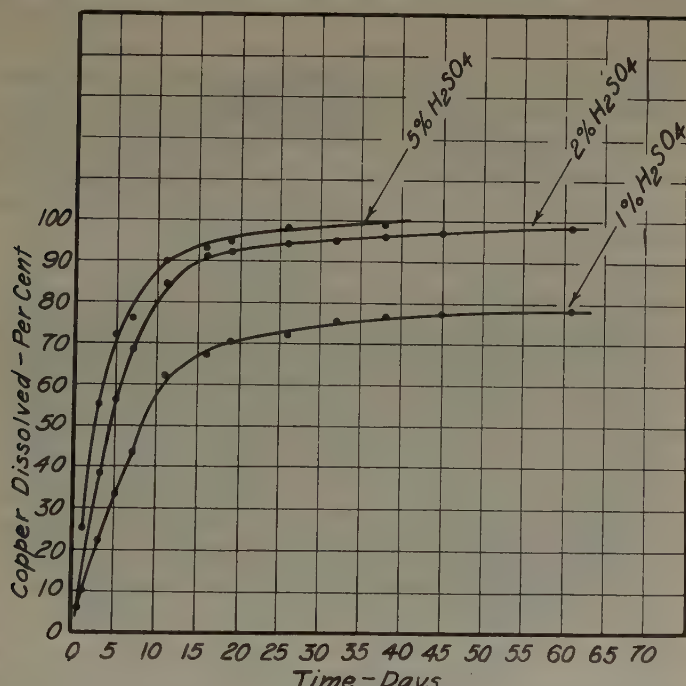

FIG. 2.—RATE OF DISSOLUTION OF MINUS 10 PLUS 28-MESH DIOPTASE IN SULFURIC ACID OF VARYING STRENGTHS.

not. Chrysocolla is a definite mineral, $CuSiO_3.2H_2O$, and is easily dissolved in sulfuric acid. Insoluble copper silicate encountered in ores may be dioptase, although there are several other copper silicates, including bisbeeite, cornuite, plancheite and shattuckite, of which the chemical properties are not definitely known.

Cuprite.—Cuprite is a mineral often considered not suitable for sulfuric acid leaching. Hofman[4] states: "Cuprite is decomposed into CuO and Cu; and ore carrying cuprite has to be oxidized (weathering,

[4] H. O. Hofman: Metallurgy of Copper (revised by C. R. Hayward), 283. New York, 1924.

roasting) before treatment." Greenawalt[5] says: "When acted upon by dilute sulfuric acid, it is partly reduced to metallic copper, and partly oxidized to sulfate."

In an earlier paper by the author[6] it was shown that cuprite can be leached by either sulfuric acid or acidified ferric sulfate. Complete dissolution, however, can be effected by sulfuric acid alone only in the presence of oxygen. In closed bottles with oxygen excluded only half of the copper was dissolved according to the reaction:

$$Cu_2O + H_2SO_4 = CuSO_4 + Cu + H_2SO_4$$

Experiments reported in the earlier paper showed that even with pieces as large as 3 mesh, all of the copper in cuprite was dissolved in ferric sulfate in eight days, and 99 per cent was extracted in three days. The time required for smaller pieces was much less. Minus 100 plus 200-mesh material was virtually all dissolved in one hour. With sulfuric acid in the presence of air the rate of dissolution was somewhat slower than with ferric sulfate, but even with the minus 4 plus 10-mesh size, 99 per cent of the copper was dissolved in 20 days, and approximately 94 per cent of the copper from minus 3 plus 4-mesh size in the same time interval. For smaller sizes the dissolution rate was much more rapid, 24 hr. being sufficient to convert 100 per cent of the copper in the minus 100 plus 200-mesh into the soluble sulfate, while approximately 99 per cent was dissolved in 12 hours.

There is a tendency for the dissolution of cuprite to take place in two stages, one-half of the cuprite to go to sulfate and the other half from Cu_2O to Cu and then the Cu to go to copper sulfate by the help of an oxidizer. The finer the material the greater the number of particles that disappear during any given interval, and the more effective the action of air in promoting the dissolution of the finely divided copper. When cuprite is leached with sulfuric acid, metallic copper forms a difficultly permeable outer layer on the surface of the particles, which tends to slow down the dissolution. Fig. 3 shows a section of a cuprite fragment approximately ¼ in. in size that was treated for one hour with a dilute solution of sulfuric acid. The surface of the mineral, when removed from the solution, was coated with a layer of soft, spongy copper, and inside this soft layer was a hard, adhesive coat of metallic copper. The hard layer was about 0.02 to 0.03 mm. thick, while the soft, spongy layer on the outside was slightly thicker. In the photomicrograph the white material on the outside portion is metallic copper and the black center is cuprite. The scratches on the white portion indicate the hard character of the copper. The polishing was done on canvas, using

[5] W. E. Greenawalt: The Hydrometallurgy of Copper, 161. New York, 1912.

[6] J. D. Sullivan and G. L. Oldright: The Dissolution of Cuprite in Sulphuric Acid and in Ferric Sulphate Solutions. U. S. Bur. Mines *Rept. of Investigations* 2967 (1929).

considerable pressure, which shows that the deposit was adhesive. A few bright spots near the center are due to light reflected from the surface from scratches made with a metallic instrument producing an actual polishing of the surface.

When cuprite is leached with acidified ferric sulfate, sulfuric acid is the principal solvent for the reaction, converting one-half of the copper to the soluble sulfate and one-half to the metallic condition. Ferric sulfate assists in this part of the reaction by diminishing the amount of metallic copper adhering to the cuprite, and after the Cu stage is reached ferric sulfate is a much more active solvent for metallic copper than is sulfuric acid plus air.

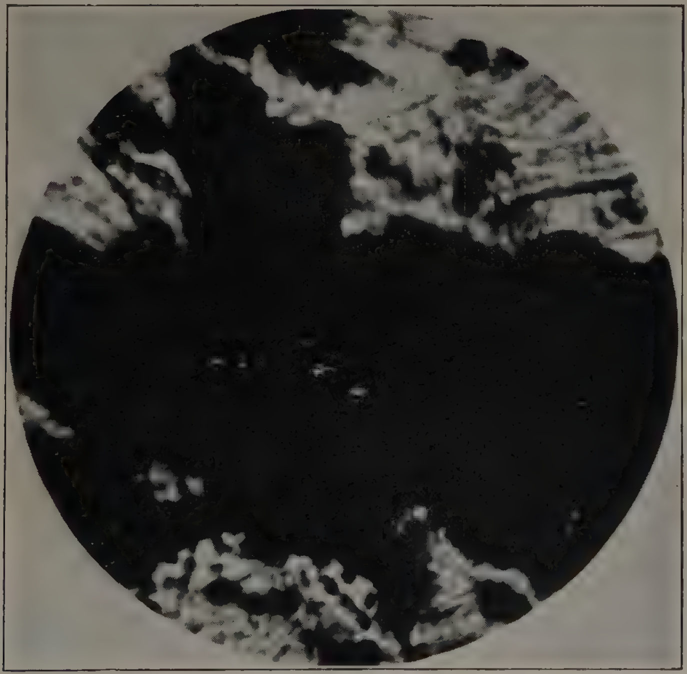

FIG. 3.—METALLIC COATING OF COPPER ON CUPRITE. × 110 APPROX.

Chalcocite.—Chalcocite is one of the most important sulfide minerals of copper to be encountered in leaching ores. In a previous paper[7] the author has given experimental details of work done on the dissolution of chalcocite in various solvents.

The rate of dissolution of chalcocite in acidified ferric sulfate solution is practically independent of the size of particles over a wide range of particle size. Thus, particles minus 10 plus 28-mesh in size dissolve at

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[7] J. D. Sullivan: Chemistry of Leaching Chalcocite. U. S. Bur. Mines *Tech. Paper* 473 (1930).

virtually the same rate as minus 200-mesh material. Pieces minus 3 plus 10-mesh in size dissolve at virtually the same rate as smaller sizes, and minus 2 plus 3-mesh pieces dissolve at only a slightly slower rate. This would indicate either that chalcocite contains in its normal conditions sufficient capillary pores or openings to permit ingress of solution, or that internal fractures are opened during the crushing process. The latter seems scarcely tenable, since the larger pieces received scarcely any breaking or crushing treatment. The rate of extraction in acidified ferric sulfate solution is virtually independent of the strengths of iron as ferric sulfate and of sulfuric acid over a wide range of concentrations. A large excess of ferrous sulfate in the leaching solvent slightly decreases the rate of extraction.

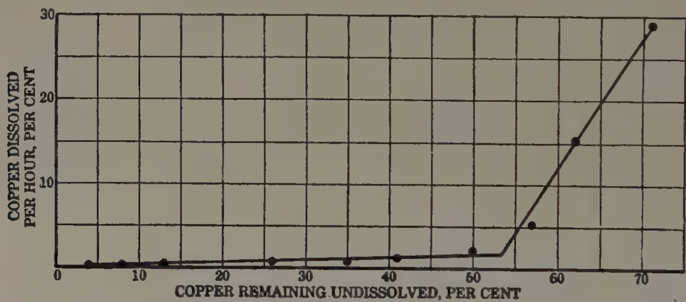

FIG. 4.—SHOWING TWO-STAGE DISSOLUTION OF CHALCOCITE IN ACIDIFIED FERRIC SULFATE.

Chalcocite definitely dissolves in two stages, the first being much more rapid than the second. For minus 100 plus 200-mesh material at 35° C., in acidified ferric sulfate solution, approximately 50 per cent of the copper is dissolved in 24 hr., and approximately 20 days is required to extract the remaining 50 per cent. In Fig. 4 data are plotted showing the two-stage dissolution of chalcocite in acidified ferric sulfate solution. The percentage of undissolved copper is plotted against the percentage of copper dissolved per hour. The curve, which is typical, shows that the reaction takes place in two steps, the first of which is much more rapid than the second. These two steps are represented by the Cu_2S rapidly giving up one-half of its copper and changing to a form approximating CuS, which dissolves at a slower rate. After half of the copper is dissolved, the material remaining, although having the approximate formula CuS, is not the same as natural covellite. After the extraction has proceeded to a certain stage, the atomic ratio of Cu:S becomes about 0.9:1.0, and the rate of dissolution is markedly different from that of natural covellite.

Pure chalcocite leached for 48 hr. at room temperature with acidified ferric sulfate, and then viewed under the microscope, contains a bluish colored mineral quite different from chalcocite and of a much paler blue

color than true natural covellite. It resembles the color often classified by mineralogists as blue chalcocite, which is claimed by many to be a mixture of white chalcocite and blue covellite, the intimate mixture of which gives a paler color than that of true covellite.

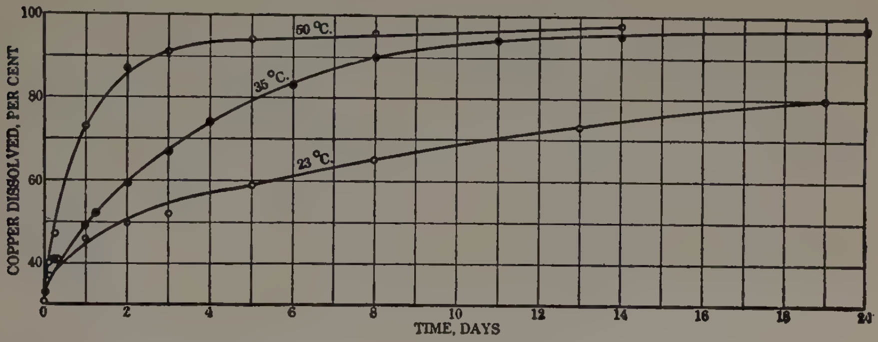

FIG. 5.—EFFECT OF TEMPERATURE ON RATE OF DISSOLUTION OF MINUS 100 PLUS 200-MESH CHALCOCITE IN ACIDIFIED FERRIC SULFATE.

Fig. 5 gives results on the rate of dissolution of minus 100 plus 200-mesh chalcocite in acidified ferric sulfate at various temperatures. Chalcocite dissolves slowly in sulfuric acid alone even in the presence of excess air. Data are given in Table 2 on dissolution rates in sulfuric acid solutions of different strengths.

TABLE 2.—*Dissolution of Minus* 100 *Plus* 200-*Mesh Chalcocite in Solutions of Sulfuric Acid of Various Strengths at* 35° *C.*

Time, Days	H_2SO_4 in 100 C.c. Leaching Solution, Grams					
	0.4375	1.75	8.75	17.5	35.0	87.5
	Copper Dissolved, Per Cent					
7	19	31	32	33	32	8
14	31	34	36	37	37	16
21	33	36	39	40	39	23
28	35	38	41	42	41	27
39	37	40	42	43	43	32

Anderson and Cameron,[8] in discussing leaching in place at the Ohio Copper Co. mine, state: "Chalcocite, Cu_2S, is quite rapidly attacked by oxygen at ordinary temperatures in the presence of water, and whatever reactions may intervene, the result may be summarized in the equation $Cu_2S + 5O + H_2O = CuSO_4 + Cu(OH)_2$."

[8] A. E. Anderson and F. K. Cameron: Recovery of Copper by Leaching, Ohio Copper Company of Utah. *Trans.* A.I.M.E. (1926) **72,** 31.

Experiments were made in which minus 100 plus 200-mesh particles of chalcocite were submerged in water with a steady stream of air passing over the surface of mineral particles. After 35 days only 0.13 per cent of the total copper was rendered soluble. These tests indicate that air and water alone are not sufficient to extract the copper from chalcocite. It may be possible, by employing a system of alternate wetting and drying, to oxidize the surface of the mineral and thus very slowly effect a conversion of the copper to the soluble form. However, the dissolution of chalcocite by heap or in-place leaching is undoubtedly due to the presence of ferric salts and not to natural weathering agencies.

Experimental work has been reported on the mechanism of dissolution of chalcocite in acidified ferric sulfate solution.[9] The reactions may be written:

$$Cu_2S + Fe_2(SO_4)_3 = CuS + CuSO_4 + 2FeSO_4$$

$$CuS + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4 + S$$

Or the combined reactions may be written:

$$Cu_2S + 2Fe_2(SO_4)_3 = 2CuSO_4 + 4FeSO_4 + S$$

Figs. 6, 7 and 8 show minus 2 plus 3-mesh chalcocite before leaching, after leaching for 14 days at 50° C., and the residue after treatment with carbon bisulfide. After leaching the particles retained nearly their original outline. Apparently the sulfur remaining behind retains the form of the original mineral. If carbon bisulfide is added, the sulfur dissolves and the particles collapse.

The rate of dissolution of chalcocite at ordinary temperatures is not markedly different in ferric chloride and ferric sulfate solutions. Attack by ferric chloride is somewhat more rapid at boiling temperatures. Slowly trickling a solution of ferric sulfate through a column of chalcocite effected a somewhat slower rate of dissolution than in the rotating bottles. Whereas the percentage dissolution was 47 and 90 in 1 and 11 days, respectively, in bottles, it was 33 and 79 in the same time, respectively, in the trickle tests.

Bornite.—The pure mineral bornite has the chemical formula Cu_5FeS_4. Bornite is rarely pure and is frequently associated with chalcocite and chalcopyrite. Smaller amounts of covellite are also frequently associated with it. The fresh fracture of bornite has a peculiar red-brown color from which it is frequently designated as "horseflesh ore."

Acid or some sulfate carrier is necessary to convert bornite into copper sulfate. Acid and air attack bornite much more slowly than does a good oxygen-carrying reagent, such as ferric sulfate. Fig. 9 gives data on the rate of dissolution of two samples of bornite and one of chalcocite in acidified ferric sulfate solution.[10] At the start the copper was extracted

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[9] Reference of footnote 7, 17–22.

[10] J. D. Sullivan: Chemistry of Leaching Bornite. U. S. Bur. Mines *Tech. Paper* 486 (1931).

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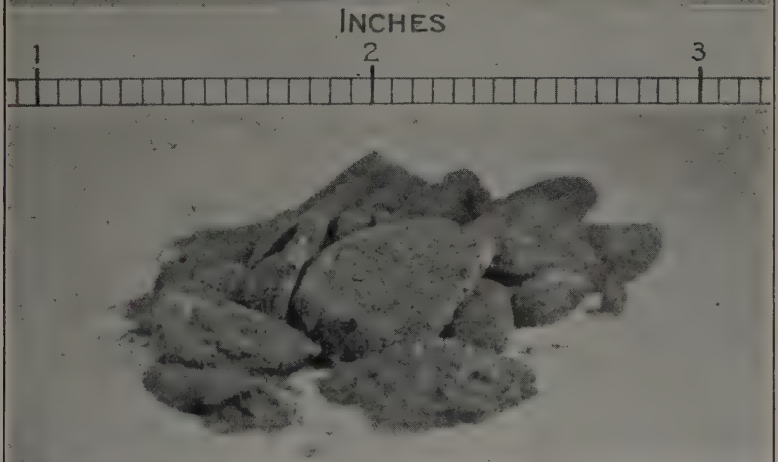

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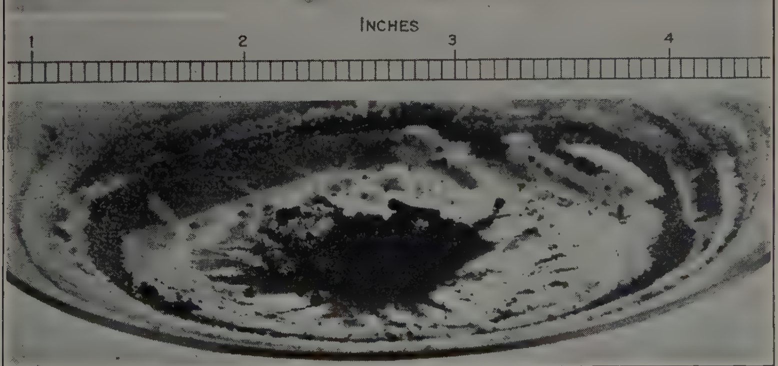

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FIG. 6.—MINUS 2 PLUS 3-MESH CHALCOCITE BEFORE LEACHING.

FIG. 7.—MINUS 2 PLUS 3-MESH CHALCOCITE AFTER LEACHING 14 DAYS AT 50° C. WITH ACIDIFIED FERRIC SULFATE.

FIG. 8.—RESIDUE FROM LEACHING MINUS 2 PLUS 3-MESH CHALCOCITE AFTER TREATING WITH CARBON BISULFIDE.

more rapidly from the chalcocite, but approximately complete extraction was effected in about equal time. The rate of extraction was markedly increased by increases in temperature, as shown in Fig. 10.

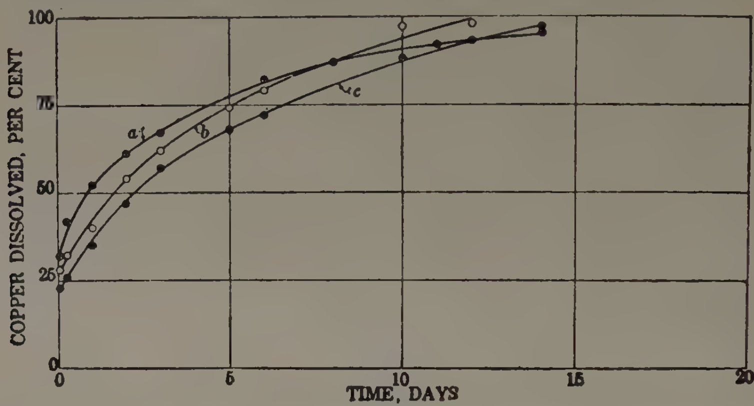

FIG. 9.—RATE OF DISSOLUTION OF MINUS 100 PLUS 200-MESH BORNITE AND CHALCOCITE IN FERRIC SULFATE. *a*, CHALCOCITE; *b*, MAGMA BORNITE; *c*, SUNSET BORNITE.

As in the case of chalcocite, the rate of extraction of copper from bornite by acidified ferric sulfate was not markedly affected by size of particles over a wide range of particle size.[11] Bornite was dissolved more rapidly by ferric chloride than ferric sulfate. At 35° C. approximately 60 per cent of the copper from minus 100 plus 200-mesh material was dissolved in ferric chloride solution in 1 day, 90 per cent in 6 days,

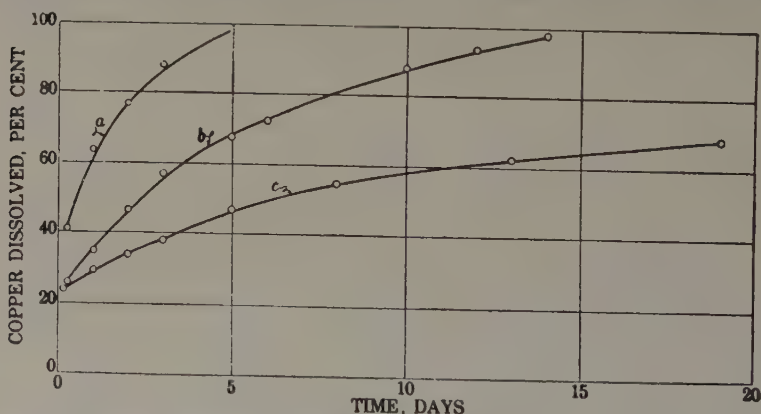

FIG. 10.—EFFECT OF TEMPERATURE ON RATE OF DISSOLUTION OF MINUS 100 PLUS 200-MESH SUNSET BORNITE IN ACIDIFIED FERRIC SULFATE. *a*, 50° C.; *b*, 35° C.; *c*, 23° C.

and about 99 per cent in 10 days. Ferric chloride acidified with hydrochloric acid dissolved copper from bornite more slowly than solution made of ferric chloride alone. A solution containing 1 per cent of iron as

[11] Reference of footnote 10, 4.

ferric chloride plus 0.5 per cent HCl dissolved 48 per cent of the copper in 1 day, and 89 per cent in 10 days. At boiling temperatures about 65 per cent was dissolved in 1 hr. and complete extraction was effected in 12 hr. in acidified ferric chloride.

Bornite was very slowly attacked by water and air alone; only about 0.1 per cent of the total copper being removed in 32 days. The mineral was more rapidly attacked by sulfuric acid; 27 per cent of the total copper was extracted in 24 days by 1 per cent H_2SO_4 and 34 per cent by 5 per cent acid. The rate of dissolution was virtually the same in acidified solutions of ferric sulfate containing 0.25 to 5.0 per cent of ferric iron. Also, the rate of dissolution was virtually the same in solutions varying from 0.25 to 10.0 per cent H_2SO_4 if the iron as ferric sulfate remained constant at 1 per cent. Dissolution was somewhat slower in tests in which solution was slowly trickled through a column of mineral than in the tests in rotating bottles. In revolving bottles the percentage dissolution was 43 and 75 in 1 and 5 days, respectively, whereas in the trickle tests the percentage was 31 and 57 in the same time intervals, respectively.

Data in Table 3 show the mechanism of dissolution of bornite in ferric sulfate solution. During the early part of the dissolution, copper was extracted, while the iron and sulfur were not appreciably attacked.

Table 3.—*Showing Mechanism of Dissolution of Bornite (Sunset) in Ferric Sulfate*

Time	Copper Dissolved, Per Cent	Iron Dissolved, Per Cent	Total Sulfur in Original Sample Existing in Residue in Form Soluble in CS_2, Per Cent	Ratio by Weight of Ferrous Iron Produced to Copper Dissolved	Ratio of Gram-atoms of Cu:Fe:S Extracted	Ratio of Gram-atoms of Cu:Fe:S Extracted during Interval Since Last Sample	Atomic Ratio Cu:Fe:S in Residue after CS_2 Treatment
Original							4.43:1.00:3.70
1 hr...	23.8	1.7	2.0	1.82	1.00:0.016:0.069		3.42:1.00:3.63
6 hr...	28.3	0.8	4.0	1.88	1.00:0.007:0.127	1.00: :0.42	3.20:1.00:3.61
24 hr...	43.5	5.8	6.9	1.83	1.00:0.030:0.135	1.00:0.074:0.15	2.64:1.00:3.59
3 days.	62.4	11.7	17.6	1.81	1.00:0.042:0.235	1.00:0.070:0.47	1.88:1.00:3.36
7 days.	77.0	35.8	45.3	1.76	1.00:0.105:0.493	1.00:0.370:1.58	1.54:1.00:2.83

As the test progressed both iron and sulfur were also appreciably extracted from the mineral. During the interval from three to seven days more gram-atoms of sulfur were liberated than copper dissolved, indicating an attack on the iron sulfide portion of the mineral. The data in the last two columns of the table, showing that the ratios of Cu:Fe:S in the residues after the free sulfur was removed by extraction with carbon bisulfide, also demonstrate the preferential attack on the copper portions of the mineral. It will be noted that, although in the original mineral the ratio of Cu:S was greater than 1:1, after leaching one hour the ratio was less than 1:1.

Microscopic examinations showed that attack to the interior of mineral fragments was very rapid. Pieces as large as 10 mesh, after being treated in ferric sulfate solution for one hour, showed a change in color to the very interior of the specimen. This is in keeping with the fact previously discussed that the rate of extraction is not markedly affected by particle size, at least over rather wide limits.

As was the case with chalcocite, large pieces after leaching retained nearly their original outline, but after the addition of carbon bisulfide the sulfur dissolved and the particles collapsed.

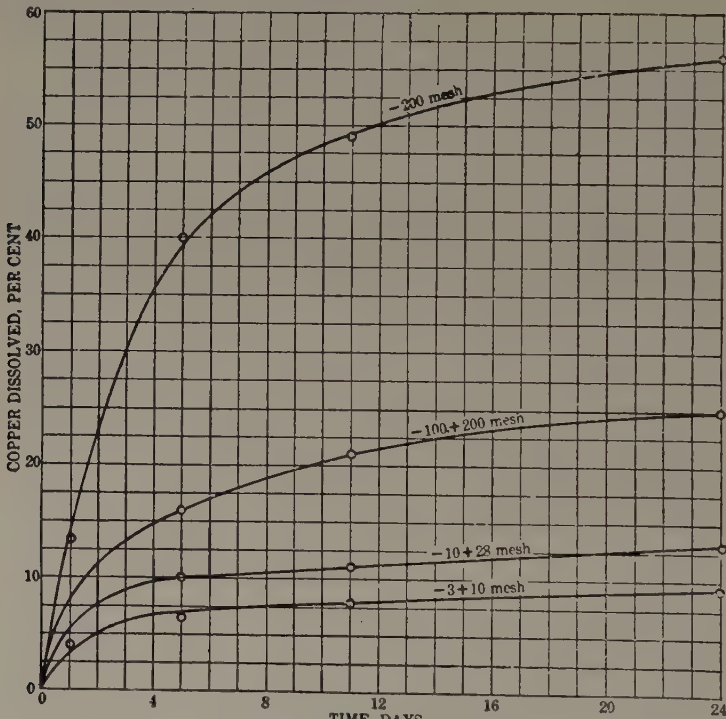

FIG. 11.—DISSOLUTION OF VARIOUS SIZES OF BUTTE COVELLITE IN ACIDIFIED FERRIC SULFATE.

Covellite.—Covellite dissolves at a much slower rate than chalcocite and bornite and different samples of covellite also dissolve at different rates. The purest samples used by the writer were from Butte, Mont. Microscopic examination showed the Butte mineral to be almost pure, containing not over 5 per cent of other copper minerals, largely chalcocite with a small amount of bornite. The sulfide impurities in the products from Kennecott, Alaska, varied from 10 to 15 per cent and consisted largely of chalcocite but with some chalcopyrite also present.

Figs. 11 and 12 show the rate of dissolution in acidified ferric sulfate of covellite of different size from two sources. It will be noticed that the

purer Butte product dissolved at a slower rate than that from Kennecott. Despite the fact that the smaller sizes dissolve more rapidly than the larger ones, the difference is not so great as would be expected from the new area of surface exposed in grinding. Apparently, covellite is also somewhat porous by nature or internal fractures capable of permitting influx of leaching solution are opened in crushing and grinding.

The rate of dissolution is increased by increases in temperature. This is shown in Fig. 13. The experiment at 98° C. was made in a beaker on a hot plate.

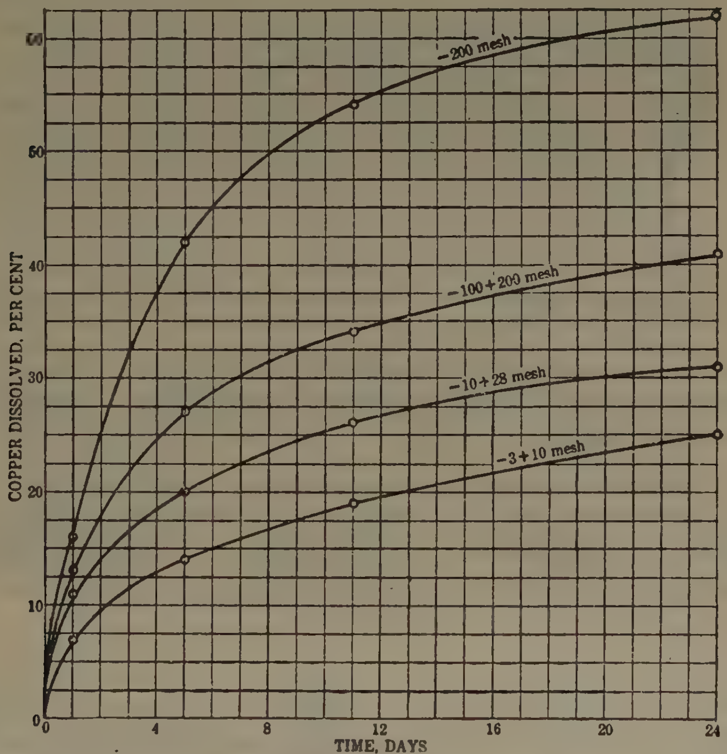

FIG. 12.—DISSOLUTION OF VARIOUS SIZES OF KENNECOTT COVELLITE IN ACIDIFIED FERRIC SULFATE.

Whereas copper is extracted from chalcocite at about the same rate by both ferric chloride and sulfate and from bornite more rapidly by ferric chloride, covellite is attacked more rapidly by ferric sulfate. Work previously reported[12] has shown that at 35° C. minus 200-mesh covellite from Kennecott when treated with a solution containing 1 per cent of iron as ferric chloride yielded 13, 22 and 34 per cent of the copper in 1, 5

[12] J. D. Sullivan: Chemistry of Leaching Covellite. U. S. Bur. Mines *Tech. Paper* 487 (1930).

and 22 days, respectively; whereas with a solution containing 1 per cent of iron as ferric sulfate, the percentage dissolved was 16, 42 and 62 in 1, 5

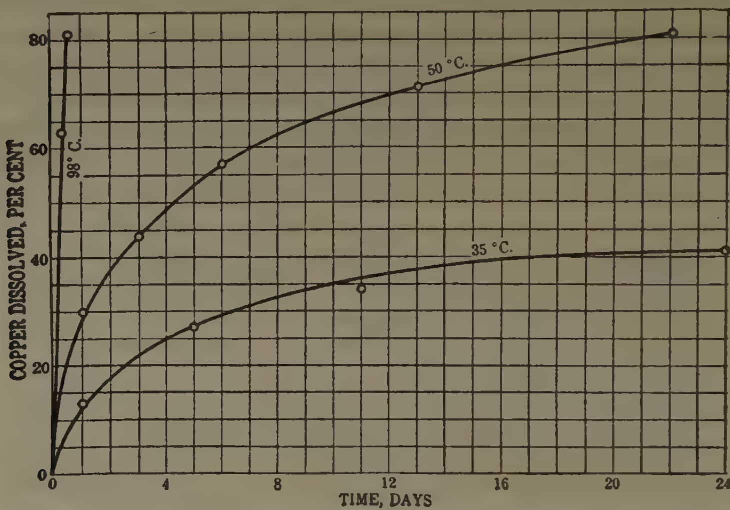

FIG. 13.—EFFECT OF TEMPERATURE ON THE RATE OF DISSOLUTION OF MINUS 100 PLUS 200-MESH KENNECOTT COVELLITE.

and 24 days, respectively. More recently Brown,[13] working on minus 200-mesh covellite from Butte, obtained similar results. Brown's data are given in Table 4.

TABLE 4.—*Dissolution of Minus 200-mesh Covellite at 35° C.*

Time, Days	1 Per Cent of Iron as $FeCl_3$	1 Per Cent of Iron as $FeCl_3$ plus 0.5 Per Cent of HCl	1 Per Cent of Iron as $Fe_2(SO_4)_3$ plus 0.5 Per Cent of H_2SO_4
1	10	10	19
3	13	13	28
5	16	17	35
8	19	20	48
16	24	26	50
22	27	28	52

Minus 200-mesh covellite (Butte) dissolved in 5 and 10 per cent solutions[14] of sulfuric acid at about one half the rate that it did in ferric sulfate solution at 35° C. The rate of dissolution is considerably slower in 1 per cent acid than in stronger solutions. In 35 days, 16, 26 and 27 per cent of the copper was extracted by 1, 5 and 10 per cent acid, respectively. Results have been reported previously[15] on the mechanism of

[13] S. L. Brown: Unpublished thesis, University of Arizona, 1931.

[14] Reference of footnote 12, 12.

[15] Reference of footnote 12, 13.

dissolution of covellite in ferric sulfate solution. Experimental evidence showed that the reaction may be written:

$$CuS + Fe_2(SO_4)_3 = CuSO_4 + 2FeSO_4 + S$$

Fig. 14 shows data on the rates of dissolution of natural covellite and a product obtained by leaching approximately 50 per cent of the copper from chalcocite, and leaving a residue with a ratio of Cu:S of very nearly 1.00:1.00. On the curve this is labeled "artificial." As shown in

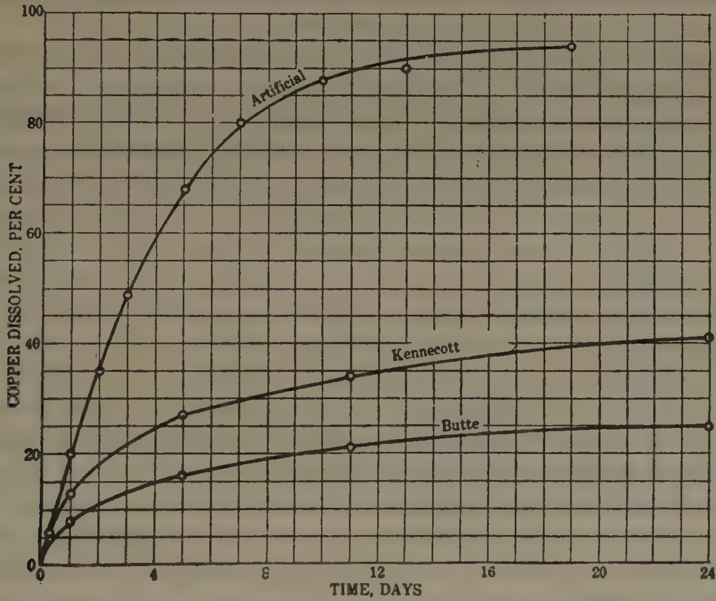

FIG. 14.—COMPARISON OF RATES OF DISSOLUTION OF ARTIFICIAL AND NATURAL COVELLITE.

Fig. 14, the artificial product dissolved much more rapidly than natural covellite. One cannot say, therefore, that when chalcocite is leached with ferric sulfate one-half the copper is rapidly dissolved leaving a residue of covellite which dissolves at a slower rate.

Chalcopyrite.—Chalcopyrite is usually considered nonleachable unless an exceedingly long time is allowed. Most of the copper may be extracted from chalcopyrite in heap-leaching processes, since the time of leaching is long and the temperature within the heaps often rises fairly high, especially if much pyrite or other sulfide is present. Chalcopyrite has not been leached successfully by vat or agitation methods, unless the ore is first roasted or hot solutions used.

Unless chalcopyrite is very finely ground, the copper is extracted very slowly at ordinary temperatures. With minus 100 plus 200-mesh material (Table 1) only 2 per cent of the copper was extracted by acidified

ferric sulfate in 43 days. With minus 350-mesh material the rate of extraction was considerably faster, the percentage extraction varying from 23 to 28 in 25 days and 33 to 39 in 57 days. The rate of extraction was accelerated by increases in temperature. Whereas only 16 per cent of the copper was extracted from the minus 350-mesh size in 14 days at 35° C., 44 per cent was removed in the same time at 50° C. About 5 per cent of the copper was extracted from the minus 100 plus 200-mesh size in 3 hr. in a boiling solution of ferric sulfate. In 63 hr., 39 and 45 per cent of the copper were extracted by boiling solutions containing 2 and 5 per cent of iron as ferric sulfate, respectively.

Copper is extracted from chalcopyrite more rapidly by ferric chloride than by ferric sulfate. At 35° C. a solution containing 2 per cent of iron as ferric sulfate extracted 11, 19, 28 and 39 per cent of copper from the minus 350-mesh size in 7, 14, 25 and 57 days, respectively, whereas one with 2 per cent of iron as ferric chloride extracted 17, 23, 32 and 45 per cent in the same time intervals, respectively. A solution containing 1 per cent of iron as ferric chloride extracted 41 per cent of the copper in 63 hr. at boiling temperature, while one containing 10 per cent of iron as ferric chloride extracted 79 per cent in the same time. Tests at boiling temperatures were made in glass beakers.

Pike, West, et al[16] stated that ferric chloride is a more efficacious leaching solvent for chalcopyrite than ferric sulfate. They concluded that 85 to 90 per cent of the copper could be extracted from Walker mine chalcopyrite concentrates by ferric chloride solutions in 2 to 3 hr. They prescribed the following conditions for successful leaching: (1) Concentration of total iron, 10 per cent or over; (2) concentration of ferric iron at least 4 per cent; (3) boiling temperature; (4) fine grinding and efficient agitation.

Data are given in Table 5 on the mechanism of dissolution of chalcopyrite in acidified ferric sulfate solution at 35° C. The mineral was crushed to minus 350-mesh size. The data in the fourth column show that the ratio of Cu:Fe:S remained virtually constant, from which one may conclude that there was no selective dissolution of any part of the chalcopyrite molecule. Since the ratio of Cu:S in the residue remained constant, and since more copper was extracted than free sulfur liberated, some of the sulfur undoubtedly was oxidized. The reactions can probably be written:

$$CuFeS_2 + 2Fe_2(SO_4)_3 = CuSO_4 + 5FeSO_4 + 2S$$
$$CuFeS_2 + 2Fe_2(SO_4)_3 + 2H_2O + 3O_2 = CuSO_4 + 5FeSO_4 + 2H_2SO_4$$

The data in Table 5 indicate that about 65 to 75 per cent of the copper was extracted by the first, and 25 to 35 per cent by the second reaction.

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[16] R. D. Pike, G. H. West, L. V. Steck, R. Cummings and B. P. Little: Electrolytic Iron from Sulphide Ore. *Trans.* A.I.M.E. (1930) **90**, Iron and Steel Div., 311.

Much consideration has been given to roasting of chalcopyrite concentrates in preparation for acid leaching. Discussion of roasting operations are outside of the scope of this paper and consideration of the factors involved in leaching roasted concentrates would make it unduly long. Roasting of any copper sulfide in the presence of iron sulfide is likely to produce a large amount of difficultey soluble copper ferrite. In the case of chalcopyrite where the iron and copper are molecularly linked, ferrite formation is especially bad. Roasting and acid leaching demands a cycle that will insure dissolution of ferrites at a low cost.

TABLE 5.—*Data on Mechanism of Dissolution of Chalcopyrite in Ferric Sulfate*

Time, Days	Copper Dissolved, Per Cent	Total Sulfur in Original Sample Existing in Residue in a Form Soluble in CS_2, Per Cent	Atomic Ratio of Cu:Fe:S in Residue after CS_2 Treatment
Original			1.00:1.18:2.20
1	8.2	5.4	1.00:1.17:2.27
7	14.8	10.4	1.00:1.16:2.23
14	21.8	15.8	1.00:1.17:2.25
21	25.5	19.1	1.00:1.17:2.25
42	30.1	22.6	1.00:1.17:2.17

Enargite.—While enargite is not so important as many other copper minerals, it exists in some districts. It is one of the most difficultly soluble copper minerals. Brown,[17] working with an almost pure specimen of enargite, referred to in Table 1 as enargite No. 2, has shown that only 2 per cent of the copper was extracted from minus 100 plus 200-mesh material in 60 days at 35° C., in ferric sulfate solutions, and less than 5 per cent from minus 200-mesh size. In dilute sulfuric acid, only 0.5 per cent was extracted in 30 days. Brown also showed that the ratio of Cu:As:S remained virtually the same after 60 days of leaching as in the original mineral. While very little copper was extracted, one can at least conclude from Brown's work that even though the copper was very slowly dissolved by ferric sulfate, the solvent did not attack the arsenic portion any more vigorously.

The author, working with a sample referred to in Table 1 as enargite No. 1, found that at 35° C. and with minus 100 plus 200-mesh material only 2.6 and 3.0 per cent of the copper was extracted in 146 days, by solutions containing 1 and 5 per cent of iron as ferric sulfate, respectively.

Tetrahedrite and Tennantite.—Tetrahedrite and tennantite are rarely found in a pure state. Analysis shows that even specimen samples usually differ considerably in composition from the theoretical amounts.

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[17] Reference of footnote 13

Invariably, tetrahedrite contains some arsenic and tennantite some antimony. Pure tennantite is supposed to contain 57.6 per cent of copper, 25.4 per cent of sulfur, and 17.0 per cent of arsenic. The author obtained one sample of tennantite which, although not pure, was much nearer to the theoretical composition than most specimens. It contained some copper that was readily soluble, but once the soluble material was removed the rest of the mineral was very refractory. Using a solution containing 1 per cent of iron as ferric sulfate plus 0.5 per cent of sulfuric acid and minus 100 plus 200-mesh material, 9.9 per cent of the copper was extracted in one day, but only 2.2 additional per cent in the following 55 days. With 2 per cent of iron as ferric sulfate plus 0.5 per cent of sulfuric acid, the extraction was 10.1 per cent in one day and only an additional 1.6 per cent in 55 days. One would conclude that tennantite is quite refractory to leaching by sulfuric acid or ferric sulfate.

While the author examined various specimens of tetrahedrite, none were pure enough to justify any conclusions as to the rate of dissolution. In the case of impure samples, the rate of extraction of copper was considerably more rapid than from tennantite.

Other Chemical Considerations.—In considering the chemistry of leaching, the mineral associations and the character of the gangue must be considered. Some carbonate ores of copper containing appreciable amounts of calcium or other carbonates cannot be leached economically by the acid process, owing to high reagent consumption. Also, the method of recovering the copper from the pregnant leach liquors may be a decisive factor in the choice of solvent. Since ferric iron reduces current efficiency in electrodeposition, the solutions going to the tank house should contain as small an amount of ferric salts as possible if high current efficiency is to be realized. Likewise, when scrap-iron precipitation is employed ferric salts consume iron. In leaching ores the relative solubilities and rates of dissolution of the copper minerals and impurities determine the reagent consumption. The ideal goal is to choose a solvent and leaching cycle to extract a maximum amount of copper and a minimum of impurities. Usually the ideal condition cannot be realized because the ore must be leached in a given time, and the cycle cannot be made too complicated. In heap-leaching or in-place leaching, since natural weathering is largely relied upon for the extraction of the mineral values, not so much attention is paid to the solvent as in methods of confined leaching.

The usual practice in vat leaching is to add the strongest solvent to the ore longest under treatment, whereas the newest ore gets the weakest solvent. The theory on which this practice is based is that the oldest ore under treatment already has the easily removed minerals extracted and the remaining more refractory ones require a stronger solution to effect their dissolution.

The writer has reported experiments[18] made by percolation methods on a leachable ore of the Southwest containing 83 per cent of the total copper in an acid-soluble condition. It was shown that a large percentage of the copper could be extracted from the ore much more rapidly by employing solutions of higher acidity, and that up to an extraction of 50 or 60 per cent of the total copper, the ratio of acid consumed to copper extracted did not increase markedly with increases in acid strength. Sixty per cent of the total copper was extracted in 48, 62 and 85 hr. by solutions containing 10, 5 and 1 per cent H_2SO_4, respectively. The ratio of acid consumed to copper extracted, by weight, was 3.1, 2.7 and 2.45, respectively. In extending the leaching period on the same ore to 240 hr. the percentage extractions were 85.8, 81.9 and 79.7 for the three acid strengths, respectively, and the ratio of acid consumed to copper extracted increased to 7.65, 4.87 and 2.83, respectively. Use of the 1 per cent acid over a longer period did not increase the acid consumption so much as in the case of the stronger acids. As a matter of fact, a price was paid in acid consumption for the last of the copper extracted by any of the solutions. For the last 2 or 3 per cent of copper removed the ratio of acid to copper was 10.2, 19.5 and 39.2 for 1, 5 and 10 per cent H_2SO_4, respectively. With an extraction of 79.7, 81.9 and 85.8 per cent of the copper by the 1, 5 and 10 per cent solutions, the ratio by weight of iron to copper dissolved was 0.26, 0.59 and 1.04; and the ratio by weight of Al_2O_3 to copper dissolved was 0.38, 0.61 and 0.97, respectively.

From a chemical standpoint, as regards consumption of acid in leaching mixed ores largely in the oxidized condition, it would appear desirable to first leach the ores with a strongly acidic solution to remove a large percentage of the copper, and then to complete the leaching with a solution of lower acidity.

Recirculation of a part of the solutions in heap-leaching operations is highly desirable. Solutions in passing through the heaps ordinarily pick up ferric sulfate, and the recirculation of such solutions will accelerate the extraction of the copper from the ore.

Ingress of Solutions into Ores

While leaching is primarily a chemical process, physical factors must also be considered. One of the most important factors is the ingress of leaching solution into the interior of the particles of ores where contact is made with the copper minerals which are then, by chemical action, converted to soluble salts.

There are, in general, two classes of voids in rocks: (1) fractures and cleavage planes, and (2) pores of capillary size or smaller. Crushing tends to create and enlarge fractures. Approximately 4 per cent of the

[18] J. D. Sullivan and G. L. Oldright: Leaching of Copper Ores.—Effect of Strength of Acid in Leaching Solvent. U. S. Bur. Mines *Rept. of Investigations* 3106 (1931).

total volume of typical porphyry ore of the Southwest is occupied by voids within the particles. From the various properties affecting the ingress of solutions, rocks may be classified as follows: (1) of the "shell" type, in which solutions penetrate more or less uniformly, (2) in which solutions penetrate along fractures or cleavage planes to main channels and then from these larger openings penetrate through smaller openings or pores, the rock itself weathering very little, and (3) those that break down in weathering.

In Fig. 15 results are given on the rate of penetration of water into a typical porphyry ore of the Southwest. Details of the method of

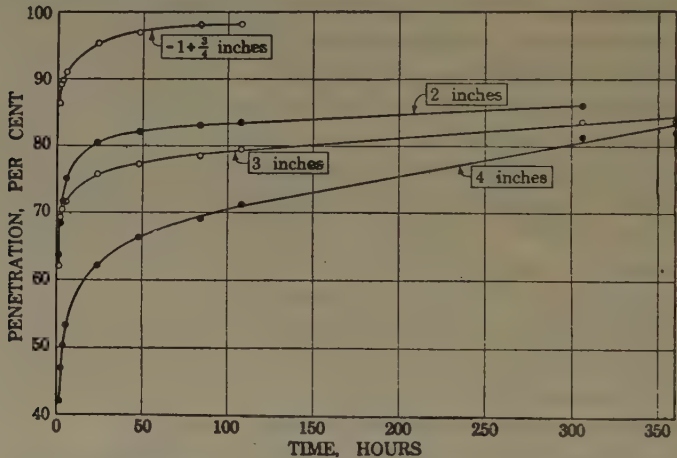

FIG. 15.—RATE OF PENETRATION OF WATER INTO VARIOUS SIZES OF PORPHYRY ORE.

studying the rate of ingress of solutions into ores have been presented elsewhere.[19] While results for pieces only as large as 4 in. across a face are presented, it can readily be seen that the time required for saturation increases rapidly with increases in particle size. Since pieces minus 1 plus ¾ in. in size are saturated rapidly, the factor of saturation is not so important in vat leaching, since particles are rarely larger than ⅜ in. However, in heap-leaching, where pieces 2 or 3 ft. across a face are common and pieces 5 to 8 ft. frequently are found, ingress may be very important. There is reason to believe that it would be economical to crush the ore before putting it on a heap. Crushing would make the ore more amenable to leaching by exposing more surface, by opening fractures and cleavage planes, and by shortening the path required for the solutions to come in contact with the copper minerals within the particles of ore. Of course, there is danger of crushing too far, since fines may be

[19] J. D. Sullivan, W. E. Keck and G. L. Oldright: Factors Governing the Entry of Solutions into Ores during Leaching. U. S. Bur. Mines *Tech. Paper* 441 (1929).

J. D. Sullivan and E. Ostrea: Factors Governing the Entry of Solution into Ores during Leaching, II. U. S. Bur. Mines *Tech. Paper* 498 (1931).

produced that will counteract the advantage of crushing by surrounding the particles of ore, by preventing free passage of solution, and by slowing down the drying process that assists in removing soluble salts from the interior of the rocks. Shortening of the path required for leaching solutions to come into contact with copper minerals and for the subsequent removal of the dissolved copper is important. Enough solution cannot enter the pores of the usual leaching ore at one time to dissolve all the copper-mineral content. Since the percentage of the total volume occupied by voids is so small, and since the concentration of copper in the solution coming out of the rock is so low, many replacements of solution are necessary to effect complete dissolution. Unless the rock breaks down in weathering, and thus shortens the path, large pieces of rock cannot be leached effectively without a very long period of operation.

The rate of entry of solutions into rocks is probably governed primarily by the solubility of gas within the voids, and the rate at which solutions wet the solid.[20] The two factors probably are interdependent, since the replacement of air in voids by a more soluble gas also permits more rapid wetting of the solid by the liquid if the air film adhering to the solid is also replaced by a more soluble gas. When the voids of a 2-in. cube of rock contained air, only 86 per cent of the voids were filled after 336 hr. of immersion in water. However, when the rock was evacuated, and the air replaced with sulfur dioxide, 86 per cent of the voids were filled in ½ hour.

Fig. 16 illustrates the manner in which solution penetrated a leaching ore of the Southwest. The sample was immersed in a solution of ferric sulfate for 3 hr., and later was immersed in a solution of sodium thiocyanate to produce a dark color suitable for photographing. The piece was then broken open without drying. The dark places in the photograph indicate zones of penetration. Penetration along fractures is especially noticeable. After solution has passed through the larger openings, it penetrates more slowly to the interior of the rock through smaller fissures and pores.

An important factor both as regards ingress of solution and dissolution of the minerals is the distribution of the copper minerals within the ore particles. If the minerals are concentrated in fractures and cleavage planes solution can more readily come in contact with them than if they are disseminated throughout the entire mass of the ore.

Removal of Soluble Salts from Leached Ores

Soluble salts must be removed after the solution has come in contact with the copper minerals and converted them to the soluble state. The

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[20] Reference of footnote 19.

mechanism of removal of soluble salts may be entirely different in heap-leaching and in vat-leaching operations.

In heap-leaching a certain quantity of water or solution is added at the surface; then another section is treated with solution, and so on. Several months may elapse before the first section is again treated with solution. Hudson and Van Arsdale[21] stressed the importance of alternate wetting and drying in the removal of soluble salts to the surface of the ore particles. Alternate wetting and drying has an important role but it is not the sole mechanism whereby soluble salts are brought to the surface of the particles of ore. In well aerated heaps the ore probably dries

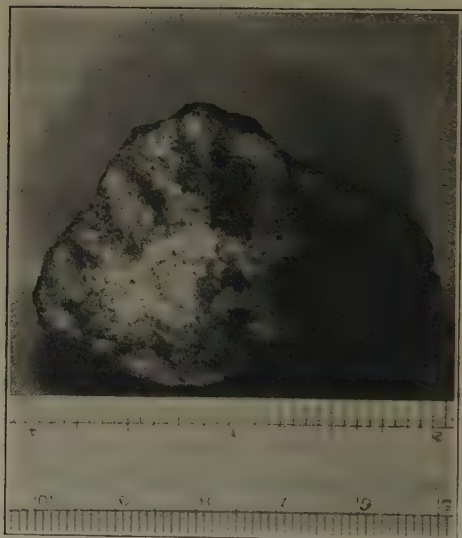

FIG. 16.—MANNER OF PENETRATION OF SOLUTION INTO ROCKS; ONE-HOUR PENETRATION.

out fairly well between the additions of solution and, during the drying, solution from the interior of the rock is brought to the surface where it evaporates and the soluble salts are crystallized. The next washing operation removes the crystallized salts. However, unless heaps are well aerated, they do not dry out. Heaps are being leached in Arizona that were not originally prepared for leaching. Waste ore was piled in dumps, and it was later decided to extract copper from them by heap-leaching. These heaps do not dry out although some drying occurs near the surface. Even in prepared heaps, the air chimneys are likely to plug owing to weathering and disintegration of rocks in the heap. Undoubtedly, diffusion plays an important role even in heap-leaching. As solution permeates the interior of the ore and dissolves copper minerals,

[21] A. W. Hudson and G. D. Van Arsdale: Heap-Leaching at Bisbee, Arizona. *Trans.* A.I.M.E. (1923) **60**, 137.

a diffusion pressure is created and soluble salts diffuse to the less concentrated solution near and at the surface of the rocks and between particles. Diffusion is a relatively rapid process and since a long time elapses between washes, considerable copper may be diffused from within the particles of ore. On the other hand, it is not essential that the ore be thoroughly dried in order to bring some soluble copper to the surface. Nor will a single complete drying bring all the soluble copper to the surface. When minus 1 plus ¾-in. pieces of ore were saturated by soaking in a dilute copper sulfate solution, only 60 to 65 per cent of the soluble copper was brought to the surface in a single complete drying of the particles and less in the case of larger pieces.[22] Fig. 17 illustrates a 6-in. piece of ore that had been saturated by immersion in a 5 per cent solution of copper sulfate, and then dried. The picture illustrates the interior of the rock which was broken after drying. The black spots and sections denote copper sulfate. The copper sulfate is disseminated through the entire rock, especially along the fractures and cleavage planes. This illustrates that in a single drying operation only part of the soluble copper is brought to the surface.

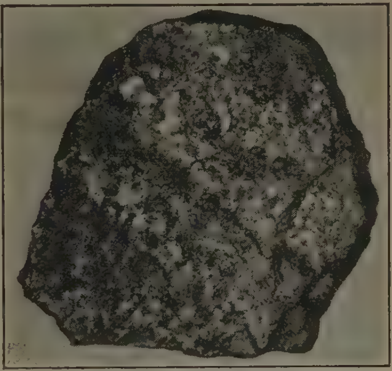

FIG. 17.—CRYSTALLIZED COPPER SULFATE IN INTERIOR OF ROCK.

Removal of soluble salts in confined leaching is due to diffusion since the ore is entirely immersed in leaching solvent and wash solutions. Most of the copper has probably diffused into the main solution before washing operations commence.

Certain physical factors are very important in washing operations. Presence of fines and slime retard washing operations. Some slimes have the power of adsorbing large amounts of copper salts, and the latter are removed with difficulty. Unless sufficient acid is present in the first wash solutions precipitation of basic salts of iron and copper is likely to occur.

An extensive series of laboratory tests on the removal of water-soluble copper salts from ores by washing have been reported.[23] These tests simulated conditions after draining off the pregnant solution from leached ores. The most important conclusions drawn from this investigation follow:

[22] M. Guggenheim and J. D. Sullivan: Acceleration of Extraction of Soluble Copper from Leached Ores. U. S. Bur. Mines *Tech. Paper* 472 (1930).

[23] J. D. Sullivan and K. O. Bayard: Extraction of Soluble Copper from Ores in Leaching by Percolation. U. S. Bur. Mines *Rept. of Investigations* 3073 (1931).

1. Ability of the ore to retain solution is a good indication of the difficulty that will be found in washing out the soluble salts. The physical nature of the rock, its porosity, degree of kaolinization and sericitization, and the amount of natural slime determine the amount of solution that will be retained, and the difficulty that will be found in washing it out. Difficultly permeable slimes may decrease the rate of washing, and ores that cause slimy hydrous precipitates to form are also more difficult to wash.

2. The rate of removal of water-soluble copper decreased with increases in the percentage of slime present. The effect of slime is illustrated in Fig. 18. A given ore was crushed to minus ¼ in. and to

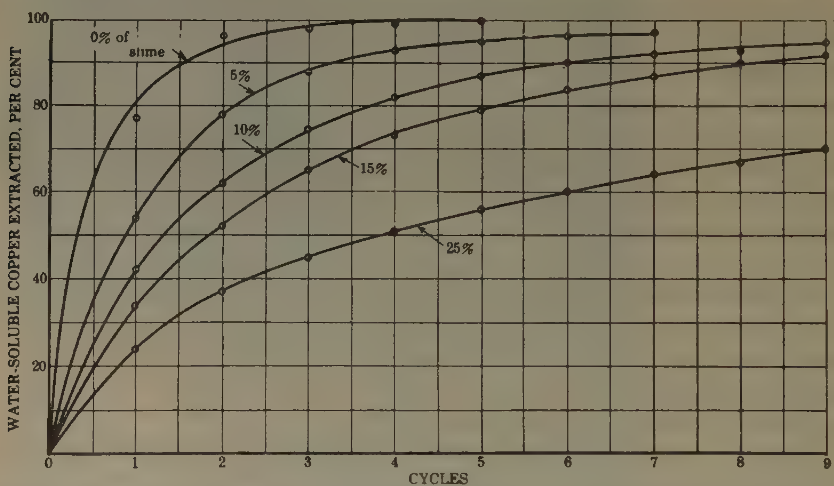

FIG. 18.—EFFECT OF VARYING AMOUNTS OF SLIME ON RATE OF EXTRACTION OF WATER-SOLUBLE COPPER.

five separate portions—0, 5, 10, 15 and 25 per cent of slime (83 per cent minus 200-mesh) was added. The ore samples were saturated by immersions in copper sulfate solution and after removal of the excess solution by draining were washed with water. The time of the washing cycle was identical in each test.

3. If the ore was moistened before charging it into the vat to aid in even distribution of coarse and fine material, and thus to prevent channeling of solution, the soluble copper was removed more readily.

4. While in general the rate of extraction of water-soluble copper was slightly accelerated with increases in temperature, there was very little difference at 3° to 5°, 25° and 50° C.

5. Complete draining of the strong pregnant solution from the ore would appear efficacious before commencing the washing operations.

6. The cycles of submergence and draining should be as close to each other as possible, consistent with fairly thorough mixing of the

copper sulfate in the wash solutions and thorough draining of the wash solution. However, if wash water is scarce, longer periods of submergence may be justified.

7. Downward and upward percolation of wash solutions gave virtually identical rates of extraction.

8. The rate of removal (percentage) of water-soluble copper was virtually the same when the ore was impregnated with solutions containing 5 to 80 grams of copper per liter, but the weight of copper removed per cycle or per unit volume of wash solution increased with increases in strength of copper in the impregnating solutions.

9. Very little advantage was gained by maintaining a head of wash solution greater than that required just to cover the ore. While the amount of copper removed in a wash was slightly increased, the amount of wash solution used increased out of proportion to the gain.

Bedding the Ore in the Leaching Vat

A very important physical factor in vat leaching is the bedding of the ore. Ordinarily an ore is crushed as fine as feasible since the rate of extraction of copper is increased with decreasing size of particles, owing to the greater surface exposed and the shorter path required for solutions to come into contact with the minerals to be dissolved. However, the greater the degree of crushing, the more fines that are produced. Care must be taken to prevent segregation of coarse and fine materials in charging the leaching vat. In the early operations at Inspiration[24] it was found that the system of bedding used at Ajo and Chuquicamata was not applicable to Inspiration ore. Later developments at Inspiration indicated the desirability of separating the primary slimes from the ore before charging it into the leaching vat. Classifiers were installed to remove the slimes and a separate slimes-leaching plant was built.[25]

For good extraction, free percolation of solution is essential, of course. If coarse and fine materials segregate in bedding, free percolation does not result. Some sections get more solution than they require; others get an insufficient amount of solution. It is not uncommon to find tailings from certain segregated areas that have a higher copper content than the ore originally charged to the vat. Free percolation has not taken place and some copper-laden solution has seeped into those areas and has not been removed by washing.

One way of eliminating the trouble from segregation is to remove the slimes and other fine materials. This is the practice at Inspiration.

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[24] H. W. Aldrich: The Inspiration Leaching Plant. Paper presented at a meeting of the Arizona chapter of the American Mining Congress, October, 1927. Published by *Min. Cong. Jnl.*

[25] H. W. Aldrich and W. G. Scott: Leaching Mixed Oxide and Sulphide Copper at Inspiration. *Eng. & Min. Jnl.* (1929) **128**, 612.

It effectively eliminates the segregation difficulties but necessitates a separate leaching unit for the slimes. Another method of preventing segregation of coarse and fine materials in bedding lies in wetting the ore before charging, so that the fine particles stick to the coarse ones. This method of charging or bedding may be termed "wet charging." The ore is prepared for wet charging by adding about 6 per cent of moisture, preferably while giving the ore a rolling or tumbling motion. Wetting can also be done by sprinkling or dripping solution on to the ore on a moving conveyor or belt. Moisture preferably should be supplied in the form of leaching solvent, since this does not add water to the wrong end of the leaching cycle, but if, on account of corrosion and other factors, leaching solution cannot be employed, water may be used. As a matter of fact, less water will be so added than by a wet classification system used in removing slimes. Another advantage of adding leaching solvent is that it commences to act as soon as it touches the ore. The wetting action, especially if in combination with a tumbling or trundling motion, causes the fine particles to adhere to the coarser ones. The ore may then be charged into a leaching vat without danger of segregation.

Laboratory experiments described in detail elsewhere[26] showed the advantage of wet-charging methods. In the case of one ore crushed to minus 3/8 in. with a leaching period of five days, the percentage of copper extracted was 57.7 when the ore was dry-charged, and 77.2 when wet-charged. In the case of dry charging, care was taken to prevent segregation by carefully spreading the ore uniformly in the vat. Conditions of temperature, strength of leaching solution, etc., were the same. In another parallel test the minus 100-mesh material was removed by dry screening and then charged into the vat. The percentage extraction in the same period was 73.5. In a longer period of leaching the percentage extractions were 75.5, 89.4 and 89.7 for dry-charged, dry-charged with minus 100-mesh material removed, and wet-charged, respectively.

In the case of another ore crushed to minus 3/8 in. the percentage extractions were 64.3, 74.0 and 78.3 for dry-charged, wet-charged, and dry-charged with minus 100-mesh material removed, respectively.

With a third ore crushed to minus 1/4 in. the percentage extractions were 50.1, 60.3 and 56.8, for dry-charged, wet-charged and dry-charged with minus 100-mesh material removed, respectively. With a fourth ore crushed to minus 1/4 in. for the same conditions of charging, the percentage extractions were 69.9, 84.9 and 82.0, respectively.

The extraction by wet-charging methods was markedly greater than by dry charging, and in three of the four ores cited, the wet-charging

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[26] J. D. Sullivan and A. P. Towne: Leaching Copper Ores—Advantages of Wet-charging. U. S. Bur. Mines *Rept. of Investigations* 3050 (1931).

method showed better extraction than that by the dry-charged ore with minus 100-mesh material removed. The simplicity of preparing the ore and the great advantage in extraction would make it seem desirable for hydrometallurgists to give careful consideration to wet-charging methods.

Leaching of Ores and Tailings Containing Large Amounts of Slimes

As an extension of the scheme of wet charging, one may consider the leaching of concentrator tailings. Extensive tailings dumps that contain enormous tonnages of copper have accumulated. The tailings before the era of flotation were usually fairly high in their copper content. Current copper-concentrator tailings are virtually all minus 65 mesh and usually contain at least 50 per cent of minus 200-mesh material. In many instances the tailings from table and vanner concentrators of former days contained nearly 50 per cent of minus 200-mesh material. Concentrator-tailings dumps frequently contain two products—sand and slime. These products tended to segregate when sent to the tailings dump, since the slimes settled more slowly than the coarser material. In some dumps certain areas consist almost wholly of sands, and others almost exclusively of slimes. The copper is usually partly oxidized and partly sulfide. Owing to the high slime content, the low copper content, and the presence of sulfide minerals, agitation leaching has not been considered feasible. The tailings are too finely divided for standard percolation.

Agglomeration offers a possible solution to the problem of leaching tailings. Agglomeration is based on the discovery that if a small quantity of water or leaching solvent is added to concentrator tailings or finely divided ore and at the same time the mass is given a rolling or shaking movement, the mass becomes agglomerated into balls generally uniform in shape. It was found that if the agglomerated mass was charged into a column and leaching solution was added by the open-drainage trickle principle, so that the material did not become flooded, the solution percolated freely and satisfactory leaching results were obtained. Surface tension probably holds the particles together and, if the leaching solvent is added slowly so that a film of solution surrounds the particles, the fine material is not washed away and the mass will not collapse under certain conditions of leaching.

Investigations on copper tailings and ores showed that sand and slime alone should not be agglomerated but should be diluted with a coarse ore crushed to about minus ¼ in. The results indicated that not over 30 or 35 per cent of minus 200-mesh material should be present in the agglomerated mass. The amount of moisture to use in agglomerating ore is rather critical. For porphyritic copper ores containing 30 per

cent of minus 200-mesh material the moisture should be about 8 to 12 per cent.

Microscopic investigations showed that the agglomerations were of two types, the first in which particles of 1 mm. or larger in size formed the core and were surrounded by a shell of finely divided material, and the

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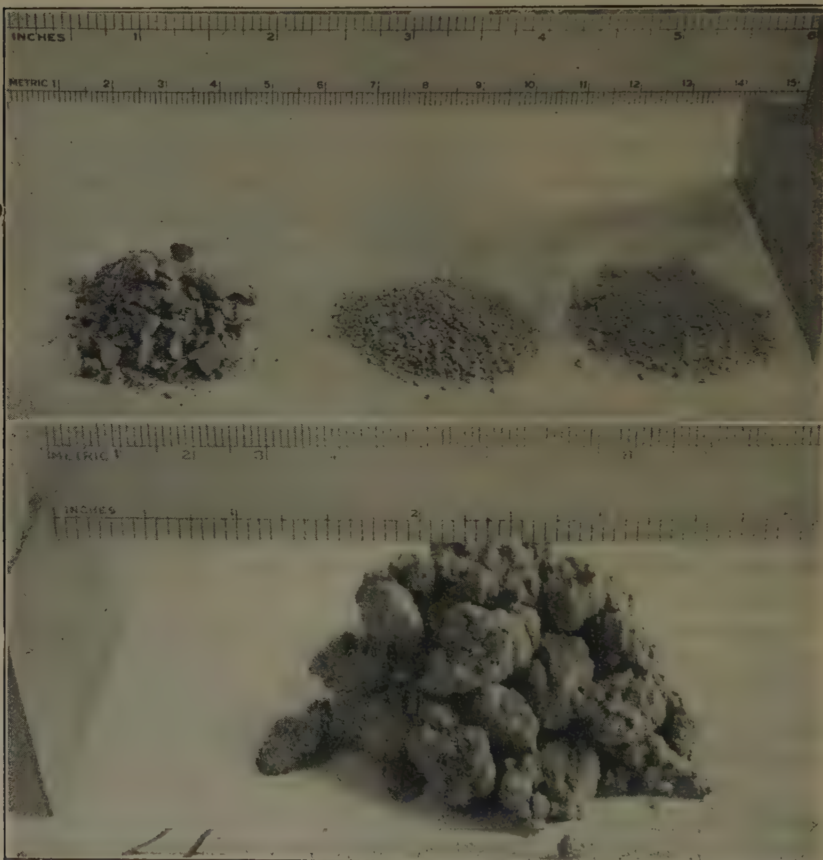

Fig. 19.—Minus ¼-in. ore, sand and slime, before agglomeration.
Fig. 20.—Same materials as in Fig. 19, after agglomeration.

second in which the particles were composed of granular members but having no clearly defined core. Further examination showed that agglomerations of the first type were units in themselves and did not appear to consolidate or attach themselves to other agglomerations except when moist. The first kind of particles is more stable and is to be preferred in the leaching process. Agglomerated copper ore has a much larger volume per unit weight than dry material. Agglomerated copper ore containing 30 per cent of minus 200-mesh material occupies approximately 28 to 30 cu. ft. per ton. Even though the agglomerated particles

in general retain their original shape during leaching, the mass frequently settles as much as 25 per cent. Figs. 19 and 20 show the nature of agglomerated particles.

In the laboratory tests the best procedure to agglomerate coarse material and tailings was to charge the coarse material and the agglomerating liquid into a revolving concrete mixer, and then after the ore was thoroughly wet and mixed, to add the tailings.

Solution must be trickled on to the agglomerated ore slowly, especially at the start of the leaching operation, to prevent flooding the ore. Flooding may cause the agglomerated mass to collapse, thus preventing free percolation of solutions. In a 12-ton test, after the mass became saturated, solution was added at the rate of approximately 3 gal. per 24 hr. per square foot of surface.

Since agglomerated ore is in an unstable condition and can be collapsed by pressure, the height of column of ore that may be used is limited. In general, the less slime present and the smaller the cross-sectional area, the higher a column is permissible. The results of laboratory experiments indicate that a depth of bed of agglomerated ore of 10 ft. or higher can be maintained.

The following is a comparison of the rates of percolation in agglomerated and unagglomerated ore and may be taken as a fair example of the advantages of the method. The two kinds of ore containing 29 per cent of minus 200-mesh material were charged into 5-ft. glass columns 1½ in. in diameter. The solution came through the agglomerated ore in 4 hr. and 15 min., and the solution did not come through the bottom of the unagglomerated ore in four days.

In leaching ores by flood-percolation methods, especially when much slime is present, considerable channeling results, whereas in trickle leaching of agglomerated ore if the solution is added uniformly at the surface there is little tendency to channel. Standard methods of leaching ore containing a high percentage of slimes present the problem of colloidal effluent solutions requiring counter-current washing or filtering before the dissolved copper is precipitated. When the agglomeration process is used, even when as much as 35 per cent of minus 200-mesh material is present, the effluent solutions are clear and require neither decantation nor filtration.

The conditions surrounding practical application of the agglomeration method will have to be determined for the individual material to be leached. Details of experimental procedure for copper ore have been described.[27] The number of possible practical applications will not be

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[27] H. E. Keyes: Innovations in Copper Leaching Employing Ferric Sulphate-Sulphuric Acid. U. S. Bur. Mines *Bull.* 321 (1930).

J. D. Sullivan and A. P. Towne: Agglomeration and Leaching of Slimes and Other Finely Divided Ores. U. S. Bur. Mines *Bull.* 329 (1930).

pointed out here, but the author believes that metallurgists and chemical engineers may find this method useful.

Precipitation of Copper from Leach Liquors

A fundamental study of the basic factors influencing current efficiency in the electrodeposition of copper from impure leach solutions with a view of improving on present practices would seem very desirable.

Acknowledgment

Many of the figures in this paper are reproduced from Bureau of Mines papers to which footnote references have been given. The author is also indebted to the Bureau of Mines for other data contained herein.

Ammonia Leaching at Kennecott

By E. J. Duggan,* Kennecott, Alaska

It is a coincidence that the two plants in this country for treating copper ores by ammonia leaching came into existence at almost the same time, that of the Kennecott Copper Corporation at Kennecott, Alaska, in March, 1916, and that of the Calumet & Hecla Mining Co. in July of the same year. The success of both from their very beginning attests the merits of the process under widely different sets of conditions.

As would be expected, in the time they have been operating these two pioneers in this field have greatly perfected their methods and improved their technic. The early difficulties inevitably encountered in undertakings of this nature have been largely surmounted and the process may now be considered to be well seasoned and matured. There have been no other commercial plants built in this country and but one elsewhere, the Bwana M'Kubwa in Northern Rhodesia, for the reason that this method of treatment may be employed on comparatively few ores. However, it has certain advantages, some of which are inherent to sand leaching processes generally and some to ammonia leaching in particular. Among these may be enumerated: (1) the avoidance of fine grinding; (2) the small amount of water required; (4) a high-grade product; (5) little likelihood of deleterious gangue constituents and fouling of solutions.

It is true that great progress has been made in recent years in the flotation of the ores that come within the province of ammonia leaching. Nevertheless ammonia leaching has its possibilities and these may be overlooked in the present tendency toward the use of flotation for all ores. Probably the chief requisite to its further extension to suitable ores will be a cheap source of steam. Ammonia, the only reagent required, is unlimited in supply, readily transported and the price that is likely to obtain in the future should not make its cost at all onerous.

Choice of Method for Kennecott Ores

The Kennecott ores are typical limestone replacement deposits. The milling ores have a copper content of approximately 8 per cent in a limestone dolomite gangue. About half the copper occurs as sulfide, mostly chalcocite with some covellite, and the remainder in the form of the two carbonates, malachite and azurite. A comparatively coarse grind followed

* Superintendent, Kennecott Copper Corporation.

by gravity concentration affords good recovery of the sulfides. The concentrator tailing, however, contains upwards of 1.00 per cent copper most of which is carbonate.

When the problem of treating this material first arose two possible methods were considered: one by sulfide filming and flotation; the other, leaching with ammonia; acid leaching, of course, being precluded by the solubility of the gangue. One of the principal determining factors in directing the management's efforts toward ammonia leaching was the fact that the shipping product would have a grade of about 75 per cent copper while the best grade hoped for with flotation would not exceed 30 per cent. Kennecott's isolated position and consequent high freight rates make a high grade for all shipping products of paramount importance. Another reason why leaching was preferred to flotation was the scarcity of water during the winter months.

The first experiments with ammonia on the Kennecott ore were conducted at the Federal Lead Co. laboratory at Flat River, Mo., in the winter of 1913–14. The possibilities of the process were immediately apparent and a small test plant was built on the property in the summer of 1914 to determine its feasibility on something like an operating scale.

The greatest obstacle encountered during the test work was the washing of the leached tailing. The ore is more or less absorptive of ammonia and cold water is a poor vehicle for its removal. Also, the large volumes required would render the cost of distillation prohibitive. It was known, of course, that heat would drive off the last of the ammonia, but the use of heat on ore of such low grade seemed highly impractical, especially as heat was the precipitant for any copper bound to be present with the ammonia. Regardless of this, investigations disclosed the feasibility of a wash in which low-pressure live steam was admitted directly to the ore, and this proved to be the salvation of the process.

Steps toward the construction of a plant were taken immediately. The first 300-ton unit was put in operation in March, 1916, and this was soon increased to a capacity of 800 tons per day. The plant operated continuously until 1931, when the collapse of the copper market caused the cessation or curtailment of operations generally. It has been described in two articles, both of which appeared in the *Engineering and Mining Journal*, one by H. M. Lawrence in the issue of Nov. 3, 1917, and one by the writer in the issue of Dec. 29, 1928. In the following pages will be recounted the practice current at the time the plant was shut down as well as the most important changes made since operations were first started.

Chemistry of Process

Aqua ammonia, or an aqueous solution of ammonium carbonate, is the lixiviant. Ammonia is purchased in the form of a liquor containing

approximately 30 per cent NH_3. Large quantities of CO_2 are absorbed by the leach liquors in the process, the CO_2 content usually being about 20 per cent greater by weight than that of the NH_3.

The chemistry of the process is simple enough and there is no fouling of solution or building up of deleterious salts, as in some of the hydrometallurgical processes. The ore is leached by percolation in the usual manner and the copper carbonates go into solution, forming various cuprammonium compounds. The enriched copper ammonia solution is then withdrawn from the ore and is followed by a weak solution wash, which is succeeded by a wash with live steam. The pregnant solution is boiled with steam, which drives off the NH_3 and CO_2 and precipitates the copper as the black oxide. The ammonia is condensed and returned to the circuit and the precipitate is filtered from the waste liquor and shipped.

Feed to Leaching Plant

As mentioned, before it is sent to the leaching plant the ore is given a preliminary treatment in the gravity concentrator; i. e., crushing with jaw crushers and Symons crushers and grinding to suitable size in rolls. The ore is then screened at 20 mesh, the undersize going to tables and the oversize to Hancock jigs. The tailings from the coarse tables and jigs comprise the leaching-plant head while the fine sand and slime-table tailings are treated by flotation.

Prior to the construction of the flotation plant the company engineers developed a method for treating the slime with ammonia in which the steam wash would have been applied in pressure filters. The steam and ammonia consumption per unit of copper compared favorably with that in the existing plant, but slime equipment would have been required and the initial plant cost would have been many times that of the flotation plant.

The feed to the leaching plant is dewatered to about 5 per cent moisture in Esperanzas of the chain and sprocket type, which discharge into a storage bin. The ore then travels via bucket elevator and horizontal chain drag conveyors to the leaching tanks. There are eight tanks, each 30 ft. in diameter, four of them taking a charge 15 ft. in depth and four a charge of 20 ft. They are of all steel construction, totally enclosed and vaportight. A false bottom made of discarded screens from the gravity concentrator is placed 4 in. from the tank bottom, on which is placed a filtering medium of coco matting and 8-oz. duck. The tanks are provided with mechanical excavators, which, when run in reverse, function as distributors. The feed is introduced through a door at the center of the top of the tank and the leached tailing is discharged through doors in the bottom to a system of conveyor belts that carries the leached tailings to the dump.

Leaching Process

After the tanks are filled with ore the charge is allowed to drain for about 24 hr. The drain cocks are then closed and the "first leach" solution is pumped in through the bottom. This "first leach" is pumped directly from the preceding charge and the ammonia contained therein is approximately 60 per cent saturated with copper. As this solution rises through the ore bed some of the residual water from the charge collects at the top of the rising column of solution. The first of this, containing traces of copper and ammonia, is siphoned off to waste and the next few inches, containing more ammonia and copper, is sent to the wash storage. The first leach is allowed to stand on the ore for about 12 hr. without circulation. The solution will then be enriched with copper to about three-fourths of the saturation point, or a ratio of NH_3 to Cu of about 1.3 to 1, which is the practical limit for good extraction. A portion of the solution is then pumped from the charge, and this portion becomes the "rich" or pregnant solution and is withdrawn from the leaching circuit for evaporation. This rich solution is then replaced by an equal volume of a mixture of low-grade make-up solution and enough concentrated ammonia from the evaporators to complete the extraction of the copper. Fresh ammonia is added if required. The solution or "second leach" is then circulated for 36 to 48 hr. by downward percolation, then it is pumped to the leaching tank containing the next charge of ore, on which it becomes the first leach.

As the second leach is pumped off it is followed immediately by 24 to 30 tons of wash solution. When the solution level is within a few feet of the bottom of the tank low-pressure live steam is admitted to the top. Most of the steam condenses on contact with the ore and virtually constitutes a hot water wash, which probably accounts for the fact that apparently no copper is precipitated during the steam wash. Pumping is continued until the charge is heated to the bottom, when the effluent is turned to a water-cooled surface condenser on which is maintained a low vacuum. Steaming is continued until the effluent has the desired NH_3 content, then the steam is turned off and the tailing is discharged to the waste dump. Table 1 gives a typical operating cycle for one charge.

The effluent solutions are divided in the following order, depending on their assay and the plant requirements: (1) second leach off becomes first leach next charge; (2) make-up solution; (3) wash solution.

The steam for the steam wash is regulated by hand and is held at a pressure of 0 to 3 lb. Little trouble is experienced in maintaining the pressure within this range, but to prevent a possible vacuum, which would pull in the tank top, or an overpressure, which would distort or burst the tank, safety devices are provided. For excessive pressure at 7-ft. gooseneck of 3-in. pipe full of water is connected to the top of each tank.

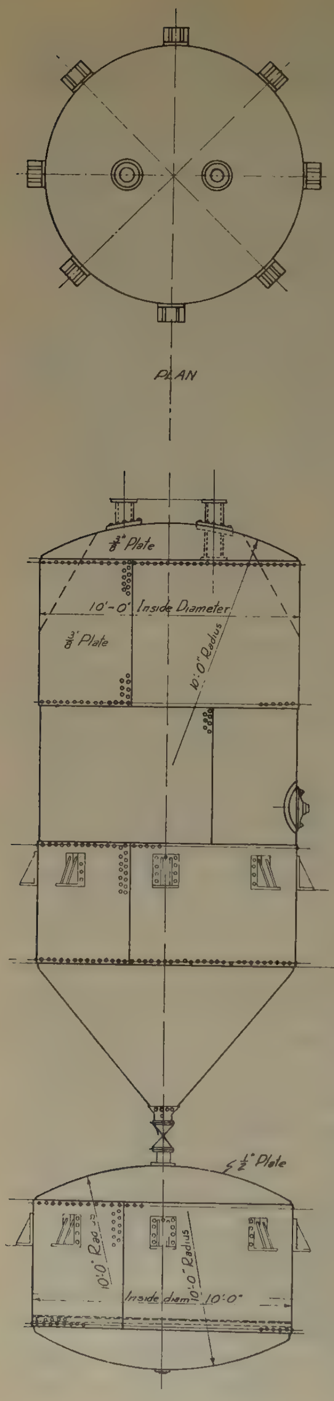
TABLE 1.—*Typical Operating Cycle for One Charge*

Hours	Operations	Amount of Solution, Tons	NH_3, Per Cent	Cu, Per Cent	CO_2, Per Cent
0– 12	Charge 575 tons ore @ 0.94 per cent carbonate Cu, 5 per cent H_2O				
12– 24	Draining				
24– 32	Pump to bottom first leach solution	192	10.50	6.30	12.40
30– 31	Siphon from top to waste	10	Tr	Tr	
31– 32	Siphon from top to wash storage	6	2.00	1.40	
32– 44	Still leach				
44– 46	Pump from bottom to rich solution storage	59	10.32	7.85	12.20
44– 46	Pump to top concentrated ammonia	23	26.00		22.00
44– 46	Pump to top make-up	36	4.00	2.50	4.30
46– 94	Circulation leach				
94–102	Pump from bottom to next charge	192	10.50	6.30	12.40
94– 96	Pump to top wash	30	1.50	1.00	
98–108	Steam wash, 28 tons steam to top				
102–104	Pump from bottom to make-up storage	36	4.00	2.50	4.30
104–108	Pump from bottom to wash storage	24	1.40	0.65	
108–120	Discharge tailing, 0.14 per cent carbonate Cu				

This automatically spills at about 3 lb. pressure. The vacuum safety gage consists of a steel box with an outlet to the leaching tank. Over perforations on its bottom rests a rubber disk held in position laterally by a pin through its center. The rubber disk rises at less than ¼ in. of vacuum and admits air to the tank. These two devices are foolproof and have never failed to function, although they are seldom called upon to perform.

EVAPORATORS

The original stills for evaporating the "rich" solution were patterned after the column stills for the recovery of ammonia in the byproducts industry and were designed to operate continuously, but the formation of hard, tenacious crusts of copper carbonate and oxide on the walls and passages of the still sections caused trouble. An excessive amount of labor was required to keep them clear and the incessant choking lowered their efficiency. Various expedients were tried, such as operating the stills in two stages, whereby the temperature at which the bulk of the copper was precipitated was under better control, but as the trouble with crusts and scale continued the present intermittent type of evaporator, similar to the one used in the experimental work, was evolved. This is essentially a 16-ft. steel cylinder 10 ft. in diameter with a dome-shaped top and a 6-ft. cone on the bottom. Heat was originally applied through



steam coils, but as the coils soon became incrusted with an insulating layer of precipitate the steam was turned directly into the solution and this ended the trouble with scale.

The evaporators may be operated in single, double, triple or quadruple effect. For many years they were operated in double effect but experiments in increasing the number of effects were so successful that triple or quadruple effect evaporation became the accepted practice. When the number of effects is increased the performance of the evaporators approaches that of the more efficient column still when the latter is functioning properly. The consumption of steam drops and the grade of the ammonia condensate is higher.

As the condensed ammonia from the evaporators is about two-thirds saturated with CO_2 it must be diluted when the NH_3 content is greater than 18 to 20 per cent, in order to prevent ammonium carbonate from crystallizing out of solution and choking the system. This is done by spraying a small amount of weak solution into the vapors entering the top of the condenser.

With sufficient tonnage to operate the evaporators at anything like capacity the best results are obtained with the quadruple effect. If the tonnage is appreciably under this amount the triple effect may be more advantageous, because there is less loss of heat by radiation. It has been found that the capacity per evaporator is approximately the same with two, three or four effects. This is approximately 26 tons per day of solution containing 10 per cent NH_3.

In the evaporators the action is as follows: As the incoming vapors heat

Fig. 1.—General assembly of ammonia evaporator and salt filter.

the fresh solution to the boiling point there is the evolution of considerable CO_2, which passes through the condenser system to a wood grid scrubber where it is washed of any NH_3 and then wasted. Next the NH_3 is driven off with the remainder of the CO_2 and the copper precipitates as the basic carbonate. Steaming continues until the charge on first effect has an NH_3 content of 0.01 per cent, by which time the copper will have been converted to the black oxide CuO. This charge is then forced by steam pressure through a salt filter, which is directly under each evaporator (Fig. 1). The filtrate is passed through settling sumps to waste and the precipitate, containing approximately 75 per cent copper, is dried by vacuum and shipped. A fresh charge of rich solution is then pumped to the evaporator and steam is turned on the next oldest charge, and they become last effect and first effect, respectively, of the next cycle.

The maximum temperatures range from 265° F. on the first effect to 220° F. on the last. The evaporators are all provided with safety valves set to blow off at 40 lb. pressure. At first these were placed on top of the dome, where they choked with ammonium carbonate in a very short time. Now they are on the side of the cylinder a short distance above the solution level and are kept clear with little trouble.

The vapors from the last effect are passed through two surface condensers in series. Rich solution is employed as the cooling medium for the first condenser while the second is cooled with water. In this manner the solutions are brought up to a temperature of 180° to 190° F. before being charged to the evaporators. Fig. 2 illustrates the arrangement and method of operating the evaporators on three effects with the preheater.

The condensate goes by gravity from the second condenser to the absorber, where it is cooled by means of cold water in lead pipe coils and is sent to storage.

Steam consumption in the evaporators varies from 0.50 to 0.60 tons of steam to 1 ton of rich solution, depending on the grade of solution, number of effects employed and the rate of capacity at which the evaporators are operated. As approximately 1 ton of solution is evaporated to 10 tons of ore, evaporation is one of the costliest steps in the process. The consumption of steam per units of ammonia and copper is much less with higher grade solutions. Hence it is imperative that dilution of the leach liquors be prevented in every way possible, as any excess water entering the circuit must be eliminated through the evaporators. Therein lies the advantage in making a high-grade ammonia condensate and in the use of a small amount of steam wash to wash the leached tailing instead of a much greater amount of fresh water.

Importance of Steam Wash

The proper application of the steam wash is the *sine qua non* in the success of the whole process. Without it the Kennecott plant probably

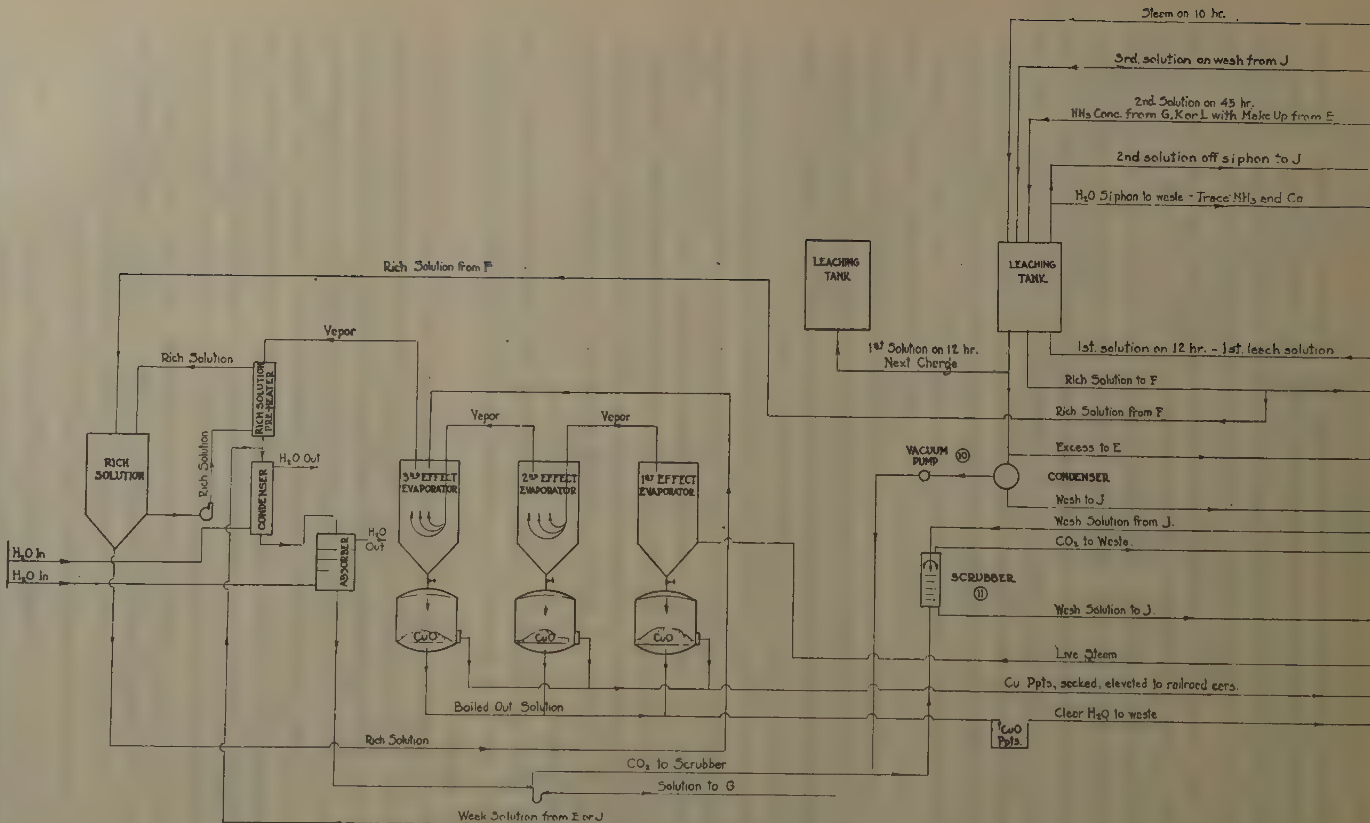


FIG. 2a.—SOLUTION FLOW.

10. Nash-Hytor No. 1 vacuum pump 11. Scrubber, 3 by 10 ft.

FIG. 2.—FLOW SHEET FOR AMMONIA LEACHING PLANT, KENNECOTT COPPER CORPORATION.

could never have operated except in periods of abnormally high copper prices. To secure good results the steaming should proceed rapidly and uniformly through the bed of ore, which necessarily demands a high degree of permeability. If precaution is taken to remove the slime and

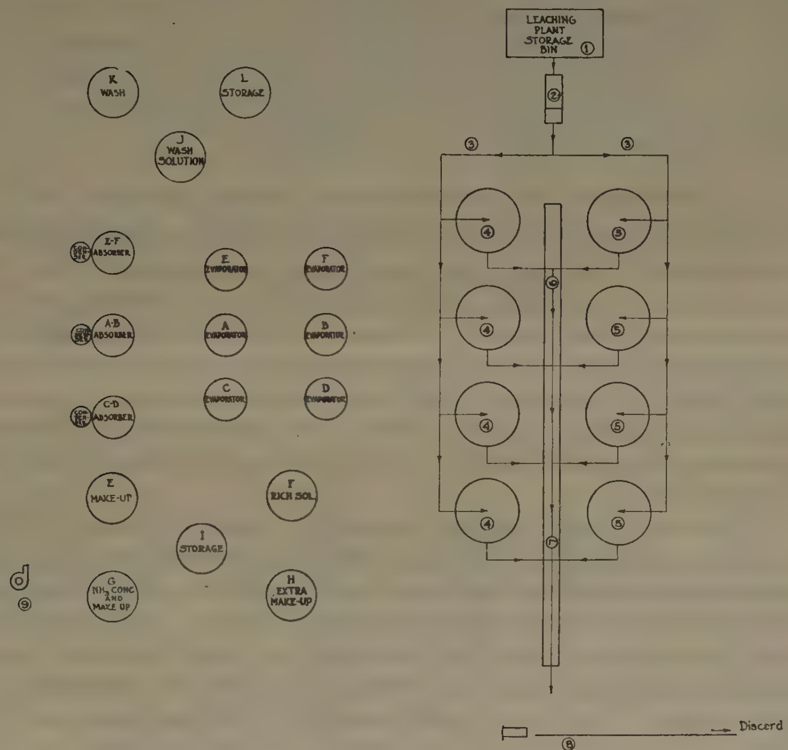


FIG. 2.—*Continued.* 2*b*.—SAND FLOW.

1. 300-ton storage bin
2. 16-in. elevator
3. Drag distributors; one 50 ft. long, two 100 ft. long
4. Four 575-ton leaching tanks
5. Four 460-ton leaching tanks
6. Two 20-in. conveyors
7. 20-in. conveyor
8. Dragline scraper for tails
9. Eight 2-in. centrifugal pumps for solution
   One 4-in. centrifugal pump for rich solution
   One 3-in. centrifugal pump for circulating water

fine sand there is no trouble with channeling. The efficacy of steam as a washing agent may be demonstrated by the fact that 1 ton of steam, following the weak solution wash, washes 20 tons of ore without the use of any other fresh water.

As originally developed at Kennecott, the steam was forced through the charge at 4 to 5 lb. pressure. The use of a vacuum, an innovation borrowed from the Calumet & Hecla plant, shortened the time for the steam wash and reduced the steam consumption about 10 per cent.

The strength of the leach liquors has been held at various times from 6 to 11 per cent NH_3 (to simplify calculations all solutions are considered to have specific gravity of unity) with corresponding grades of rich solution of 4.5 to 8 per cent copper. The CO_2 content of these solutions ranges from 7 to 13 per cent. In the last few years greater economy of steam in the steam wash and higher grades of the concentrated ammonia from the evaporators has enabled the employment of the higher grade solution. This makes for a slightly higher ammonia loss but apparently has had no effect on the extraction and but little on the steam wash. The reduction in steam consumption was considerable.

Prevention of Corrosion and Sulfur in Liquor

There is little trouble from corrosion and the original equipment that comes in contact with the leach liquors shows little evidence of it. The hot concentrated ammonia from the evaporators corrodes iron and formerly caused some trouble. This was overcome by the use of aluminum tubes and sheets in the condensers and by a concrete lining in the storage tank. Failing valves for a long time were a constant source of expense and the cause of no little loss. This difficulty was surmounted by the use of properly lubricated plug valves at all points where the service is severe.

In the early years of the plant operation, especially during and after the war, delivery of ammonia was uncertain and the cost was high. The only ammonia available at that time was from the byproducts plants and occasional lots contained a high percentage of sulfur. As the sulfur was mostly in the sulfide form, the raw liquor was mixed with the "rich" solution going to the evaporators whereby the sulfur was precipitated as copper sulfide. There were never enough of the fixed ammonia salts present to require any special provision for breaking them up. The later development of the process for production of synthetic ammonia insured an unlimited supply of high quality at a greatly reduced price.

Consumption of Ammonia and Steam

The consumption of ammonia and steam varies with the grade of the ore, that for ammonia ranging from 0.45 to 0.60 lb. NH_3 per ton of ore leached. During the last two years of operation the lower figure was attained. There is but little loss of ammonia by volatilization to the atmosphere, as is evidenced by the normally pleasant working conditions in the plant. With the evaporators operating anywhere near capacity, the steam consumption, including the steam wash, ranges from 210 to 230 lb. of steam per ton of ore approximately 55 per cent of the steam being used in evaporation and 45 per cent in the steam wash. These consumptions are also affected by the prices of ammonia, fuel and copper. As these prices do not tend to move in unison, their fluctuations may cause

some modifications in procedure with resulting changes in consumption of steam and ammonia.

Extraction and Costs

The extraction varies with the leaching time and the size of the feed. The extraction curve, however, flattens out when the time runs over 36 to 48 hr. and the dissolution of copper after that time is extremely slow. Although there is little included carbonate mineral and the values are all released at a much coarser size than that to which the ore has usually been ground, finer grinding accelerates and increases the extraction. The upper limit of size has been reduced since the plant started from 13 mm. to 9 mm. Crushing to 7 mm. was tried for a short time with apparently excellent results. Screen analyses are given in Table 2. On feed containing 0.90 per cent carbonate copper the average results shown in Table 3 have been secured, grinding at various sizes, with approximately the same leaching time. The consumption of steam for washing is affected very little by the finer feed provided the slime and finer sands are removed. The density of the feed is perceptibly less with finer grinding.

Table 2.—*Screen Analyses*

Size	Head		Tail	
	Weight, Per Cent	Carbonate Cu, Per Cent	Weight, Per Cent	Carbonate Cu, Per Cent
On 9 mm	1.62	0.46	1.12	0.09
On 6 mm	23.20	0.54	18.80	0.10
On 4 mm	26.06	0.78	26.40	0.11
On 2 mm	22.54	0.87	25.80	0.09
On 20 mesh	10.72	1.47	11.64	0.10
On 65 mesh	12.22	1.54	12.08	0.26
On 200 mesh	2.70	1.73	2.62	0.47
−200 mesh	0.94	2.00	1.54	0.76
Total	100.00	0.94	100.00	0.14

Table 3.—*Average Extraction*

Size, Mm.	Carbonate Copper in Head, Per Cent	Carbonate Copper in Tail, Per Cent	Extraction, Per Cent
13	0.90	0.22	75.9
11	0.90	0.18	80.4
9	0.90	0.14	84.8
7	0.90	0.11	88.1

The costs given in Table 4 were made when the plant was treating 19,000 tons of ore per month containing 0.92 per cent carbonate copper. During the past few years of operations, owing to general economic

conditions, the tonnage has been considerably under that figure and costs consequently were higher. More recently certain small operating economies have been made and it is believed that these costs would be improved if the plant were operating to capacity with the scale of wages and prices prevailing in, say, 1926–27, when copper was selling for 12 to 13¢ per pound. In considering costs Kennecott's location must be borne in mind. It is the terminus of the Copper River & Northwestern Railway, which extends 196 miles from Cordova on Prince William Sound into the interior of Alaska. Railroading in this region is expensive and the high cost of rail transportation, coupled with a long water haul, makes the cost of supplies abnormally high.

TABLE 4.—*Cost of Treating* 19,000 *Tons of Ore per Month*

	Cost per Ton
Operating labor	$0.080
Ammonia	0.135
Power	0.019
Steam	0.224
Repairs, labor	0.017
Repairs, supplies	0.040
Miscellaneous	0.021
Sacking and loading precipitate	0.034
Total	0.570

A Brief Description of the Reduction Plant of the Chile Exploration Company at Chuquicamata, Chile, S.A.

By T. C. Campbell,* Chuquicamata, Chile

Copper is extracted from the Chuquicamata oxide ore by a hydrometallurgical process. The ore is crushed to ⅜-in. sizing, and leached with a sulfuric acid electrolyte. Chlorine is precipitated and the ferric iron reduced in the enriched electrolyte, after which the copper is recovered by electrolysis with insoluble Chilex and lead-antimony anodes, the spent electrolyte being returned to leaching. Cathodes are melted and refined in market furnaces and cast into commercial wire-bar and cake shapes. Sulfuric acid for the process is supplied by the brochantite in the ore. Water for washing the ore is advanced through the solution system and finally after cutting by electrolysis to from 6 to 16.0 grams per liter is completely stripped by the cuprous chloride method and run to waste. The cuprous chloride so obtained plus that resulting from precipitation of chlorine from strong solution is dissolved in ferrous chloride brine and the copper cemented on scrap iron. The cement so obtained is in part used to reduce ferric iron in electrolyte, and as it is of exceptional purity is also furnace refined and cast into an exceptional quality of fire-refined copper. Fig. 1 is flow sheet of the operation.

The reduction plant is composed of seven divisions as follows: (1) crushing, (2) leaching, (3) tailings disposal, (4) dechloridizing, (5) sulfur dioxide treatment, (6) electrolytic tank house, (7) smelting and melting. A brief description of each of these divisions will be given.

Crushing

Ore that is frequently blocky and up to 5 ft. in section is crushed in four stages to ⅜-in. mesh. The first stage of crushing is called primary and the next three stages are known as secondary crushing. The first stage is accomplished with two 60-in. Superior McCully gyratory breakers by which the ore is broken to 9 in. The second coarse gyratory breakers crush the ore to 3-in. sizing. There are seven No. 10 McCully gyratories in this service, for one of which recently has been substituted a No. 7 cone crusher. The third and fourth stages of crushing are accomplished by fifty 48-in. disk crushers. In the third stage 14 disk crushers break the ore to 1 in., and in the last stage 36 disk crushers turn out the final product of 0.371-in. sizing. Reference to the flow sheets of Figs. 2 and 3

* Superintendent, Reduction Plant, Chile Exploration Co.

will show the distribution of this equipment in the new and old plants. The ore is dry and hard and crushes cleanly. A very large percentage of the ore, however, actually passes through the crushers of all four stages.

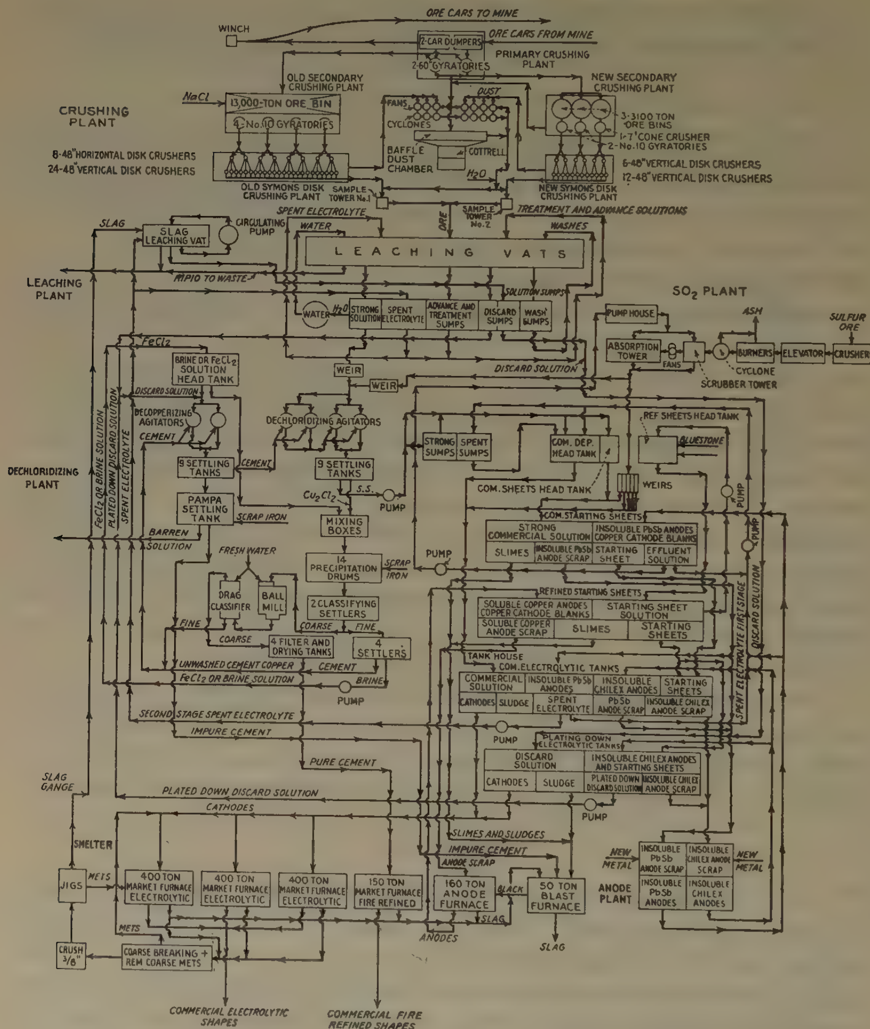


Fig. 1.—General flow sheet of reduction process, 1933.

Primary Crushers

A new primary crushing plant was completed in 1928 (Figs. 4 and 5). All of the ore crushed today passes through this plant. The building is 122 by 72 by 154 ft. high, of which 52 ft. is below ground and 102 ft. above ground level. There are five main concrete floor levels as follows:

(1) main working floor, (2) crusher motor floor, (3) eccentric floor, (4) pan feeder floor, (5) conveyor floor.

The two 60-in. gyratory breakers are set below the floor level of the main working floor. Above each crusher is a 105-ton, 30-ft. revolving car dumper. These dumpers are coupled in tandem and may be operated together or independently. Above the dumpers is a 100-ton traveling crane, and in the roof of the building is the dumper control house with

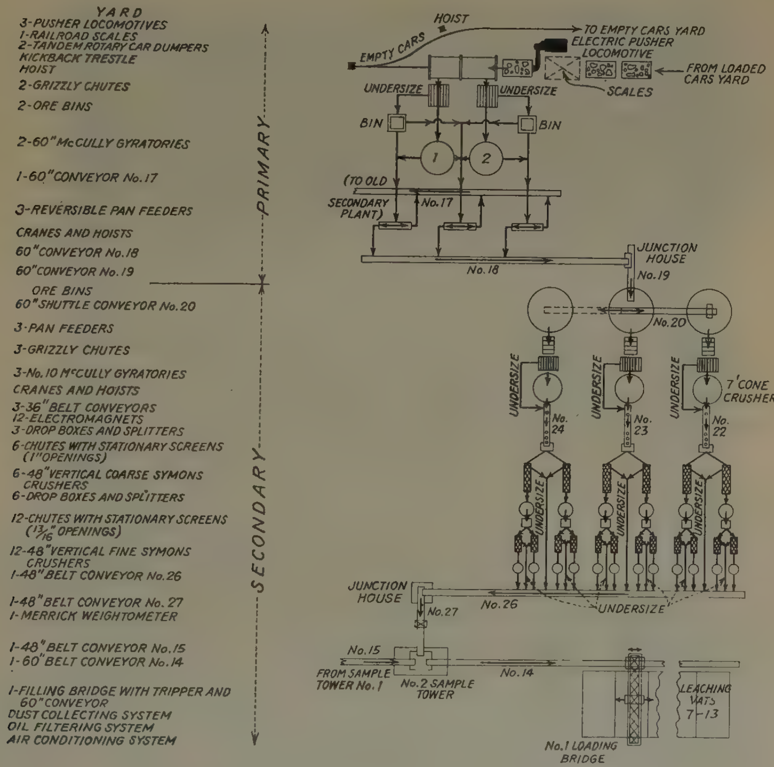


Fig. 2.—Flow sheet of east crushing plant.

glass bottom floor. A 20-ton traveling crane at the level of the car dumper floor handles the lighter crusher parts and is used also to break chokes in the gyratories. In front of each crusher there are removable floor sections in the main floor and all floors below through which eccentrics, motors, and other large parts may be removed by the overhead cranes. Small jib cranes are located at strategic points throughout. A 5-ton freight elevator handles lighter materials for the lower floors. Lubrication for the two crushers is supplied from a central station at constant pressure. Cooling equipment is installed to control the oil temperature, which is also recorded. Automatic equipment is installed

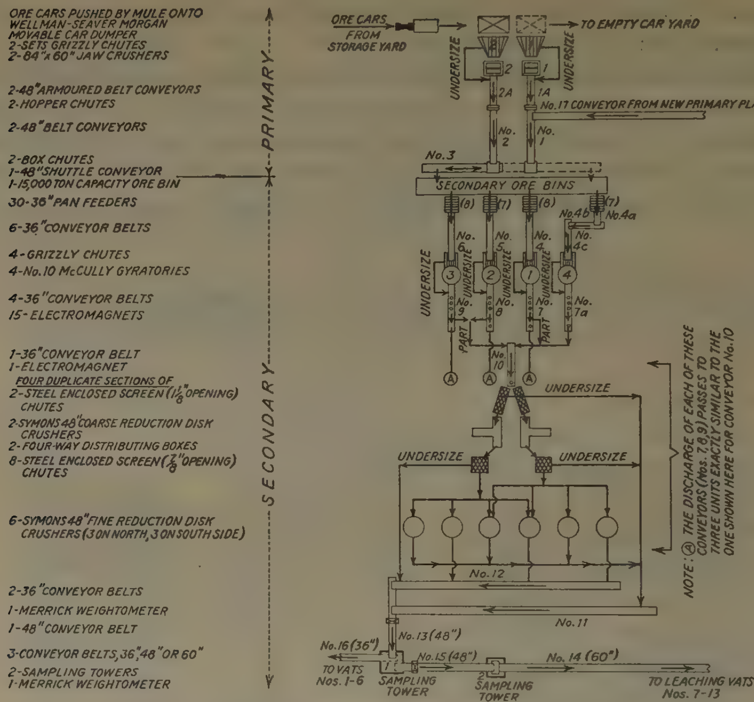


FIG. 3.—FLOW SHEET OF WEST CRUSHING PLANT.

FIG. 4.—PRIMARY CRUSHING BUILDING.

Crushing plant, substation and compressor house on the left, trestle switchback and old primary plant in background. Locomotive is one of two 75-ton battery third-rail and trolley locomotives used for plant switching.

to cut out the gyratory motors when flow of oil falls below a minimum rate. Solids in the oil are maintained below 1 per cent by bypassing to a central oil-cleaning plant, which serves the secondary plants as well and will be described further on.

Controls for motors driving the conveyors from the primary to the shuttle conveyors located on the secondary ore bins as well as the car dumpers and pan feeders are interlocked through Cutler-Hammer centrif-

FIG. 5.—SYMONS HOUSE OF NEW CRUSHING PLANT.

ugal switches on the pulley or drive shafts of this equipment. The entire equipment in this section may therefore be stopped simultaneously from any station in the section, a feature that obviates spills. All motors are of the totally enclosed dustproof type and are ventilated with conditioned air from a central station. The building is evacuated by a dust-collecting system, which serves the secondary plants as well and will be described separately. The pusher locomotives, car dumpers,

pan feeders, conveyors and other equipment in the primary plant are driven by direct-current motors, power for which is supplied from an adjacent substation, which also houses the compressors supplying air throughout the crushing plant.

Ore is delivered from the mine by electric locomotives in trains of ten 70-ton steel gondola cars to the highline of the primary crusher. This yard contains over 5000 ft. of track of 56½-in. gage and will accommodate 100 cars. Between pairs of ore-car tracks is a third standard-gage track upon which operate three pusher electric locomotives, which are used to advance the cars two at a time from the yard on to the revolving dumper. These locomotives are equipped with a cast-steel pusher arm mounted on the forward platform, operated by air and

Fig. 6.—Ore cars entering dumpers.

movable through an angle of 180° in order that cars on either side of the locomotive may be engaged. They are designed for slow speed forward and rapid reverse speed. Power is carried to these locomotives through third rail and at the crossovers at the yard head by catenary.

Before entering the dumpers the cars pass over a Fairbanks scale with Streeter-Amet automatic weighing attachment. Tare weights of cars are determined at intervals and the highline figures are used to check the conveyor weightometer recordings. As the cars enter the dumpers (Fig. 6) the empties are pushed out on to a switchback and roll by gravity into a lower yard, where they are made up into trains for return to the mine.

Each dumper delivers to a manganese steel covered grizzly 19 ft. long by 30 ft. wide with 4-in. openings at top and 8-in. at bottom, and set at angle of 33°. The sides and back of the grizzly chutes are lined with 100-lb. rail set on rubber padding. The upper ends of the grizzly bars are turned down into the ore pad to prevent large rocks from becoming embedded and hung up on the points of the grizzly bars. The oversize

ore is crushed by two 60-in. Superior McCully double discharge gyratories. These crushers are of unusually heavy design and are constructed of cast and forged steel. Because of the severe service, very ample eccentric bearing area is provided and the eccentrics are babbitted throughout. Built-in hydraulic jacks are used to lower the eccentric for inspection or removal. These jacks are operated from a fixed cylinder type of hydraulic accumulator served by a Rumsey Triplex pump operating at 400 lb. per sq. in. The jack lowers the eccentric on to a car, which can be rolled into position under the removable floor sections referred to, where it can be reached by crane. The eccentrics of these large crushers can be removed or installed in a shorter time than is required for a similar operation with the No. 10 gyratories. Each crusher is direct-driven through flexible couplings by a 500-hp. slip-ring induction motor controlled by automatic starting equipment.

The crushed ore of 9-in. sizing and the undersize from the grizzlies pass to three bins. The two outside bins take the discharge respectively from one side of each of the crushers and one-fourth of the discharge from each of the two undersize grizzly bins. The middle bin takes the discharge from the inner side of each gyratory and also takes one-half of the discharge from each of the undersize bins. These bins are equipped with shut-off gates operated from the accumulator, by means of which the feeders may be isolated for inspection or repair.

The three bins are served by three 72-in. pan feeders 34 ft. long and driven in either direction through Nuttall speed reducers and 30-hp. d.c. motors. These feeders are of the enclosed type and each has a capacity of 2000 tons per hour delivering to two 60-in. conveyors, one on either end of the feeders that carry the ore to the ground level, one to the east and the other to the west of the primary crusher building. When operating the two crushers simultaneously, the center bin and feeder take approximately one-half of the load and the other two the other half. As the feeders are of the same capacity, this difference in load is taken care of by variation of the feeder speed. From a practical standpoint the load per feeder is varied, being distributed in whatever proportion is required as determined by the condition of the bins of the secondary plants. Operation in this manner is made possible by the generous overcapacity of the feeders. Fines and dust from the pan feeders drop to three hoppers serviced by three 36-in. conveyors driven off the feeder sprocket. These conveyors deliver to the two 60-in. conveyors mentioned. At ground level the drive end of these conveyors terminates in two junction houses, where the ore is picked up by conveyors that carry it to the bins of the new and old secondary plants.

This primary crushing plant has a capacity of 70 cars of 70 tons each per operating hour. Secondary bin capacity is small in relation to plant capacity, and no other storage is employed. The bin capacity is sufficient

to take up the small difference in capacity between the primary and secondary sections of the crushing plant, and the bins may therefore be considered only as buffers.

Both plants at full production operate 16 hr. on two shifts per day, the third shift being used for clean-up and for maintenance of equipment. The working shifts for the primary and secondary plants are staggered a few hours.

FIG. 7.—SECONDARY GYRATORIES OF NEW PLANT; 7-IN. CONE CRUSHER IN FOREGROUND. NOTE DIFFERENCE IN FEEDER DESIGN.

Secondary Crushers (New Plant)

The ore of 9-in. sizing is delivered by shuttle conveyor to three silo type ore bins of reinforced concrete construction and lined with worn conveyor belting. There are three crushing sections in the new secondary plant, each serviced by a separate storage bin and composed of one No. 10 McCully gyratory crusher or one No. 7 cone crusher, two 48-in. vertical coarse disk crushers and four 48-in. vertical fine disk crushers (Fig. 7). This layout is somewhat different from that in the old plant,

where there are four crushing sections serviced by a steel and timber construction, concrete-lined, 13,000-ton ore bin, with 30 feeders and four main and three shuttle conveyors arranged for flexible drawing of the ore. Each section in this plant is composed of one No. 10 McCully gyratory, two 48-in. horizontal disk crushers and six vertical disk crushers.

Three 72-in. pan feeders are located under the new plant bins and deliver ore directly to the secondary breakers. The pan feeders are truck-mounted and may be moved clear of the bins for repairs. They are driven through speed reducers and fast flexible couplings by three totally enclosed shunt-wound d.c. motors of 15 hp. each. Each pan feeder discharges on to an 11 by 6-ft. grizzly with manganese steel bars spaced 1½ in. at the top and 2½ in. at the bottom. Two sections of this plant are equipped with No. 10 McCully gyratories and the third with a No. 7 cone crusher. The feeder for this latter type must be arranged somewhat differently and as a result the crusher is not as accessible for repairs as are the gyratories.

The undersize from each grizzly and the gyratory discharge of 3-in. sizing drops to a 36-in. inclined conveyor, which carries the ore to the disk crushers building. Electromagnets hung over these conveyors remove magnetic tramp iron. Each conveyor discharges to a splitter box from which the ore is taken by two chutes equipped with stationary cast manganese steel screens with 1-in. openings. The oversize from each chute passes to a 48-in. vertical coarse disk crusher, which makes a 1-in. product. The discharge from each crusher plus the screen undersize passes by gravity chutes to a splitter box. Two chutes equipped with manganese screens with 1³⁄₁₆-in. diameter openings take the ore from each splitter box and deliver the oversize to the 48-in. vertical disk final fine crushers. The product of these machines of ⅜-in. sizing is joined by the screen undersize and drops to a 48-in. underground conveyor. This conveyor carries the ore to an underground junction station, where it is transferred to another conveyor by which it is carried overground and to the top of the sample tower. En route collected dust is picked up and the conveyor passes over a Merrick weightometer.

Data on crushing are shown in Table 1.

In the sampling tower a bucket cutter just beyond the head of the conveyor takes a 42-ton sample per 1000 tons from the ore stream. Rolls in the tower crush this to ⅛ in., after which it is cut down on a slotted belt sampler, the reject joining the main ore stream. The sample is then sacked and transferred to the sampling division, where it is further milled, cut down and prepared as a pulp sample for assaying.

The ore after leaving the sampling tower is deposited on a 60-in. conveyor 1257 ft. long center to center, which parallels the seven east leaching vats. This conveyor runs over the head pulley of a traveling loading bridge, which spans the vats and is transferred to the lateral

conveyor of the bridge from which it is tripped by a traveling mule and discharged into the vat below. A similar sampling tower, conveyors, and loading bridge serve the old crushing plant and six leaching vats to the west. The two towers are connected by a 48-in. conveyor so that ore from the old plant may be routed to the east vats. Data on conveyor belts are shown in Table 2.

All drives in the new plant are direct through speed reducers by totally enclosed motors ventilated with conditioned air from a central air-washing plant. Conveyor and feeder motors are direct current and are interlocked from the loading bridge to the pan feeders through rotary switches in the same manner as described for the primary plant. Variable-speed conveyors and feeders are highly satisfactory, contributing immensely to the flexibility of the plant.

TABLE 1.—*Crushing Data*

Primary crushers capacity, short ton per operating hour:	
maximum	6100
average	3300
Secondary crushers capacity, short ton per operating hour:	
east plant	1200
west plant	1900
Power consumption, all equipment primary crushers, kw-hr. per short ton	0.30
Power consumption, all equipment secondary crushers, kw-hr. per short ton	1.25
Moisture in ore, per cent	1.50

The lubricating oil for the crushing machinery is supplied from a central constant-pressure system with sight feeds. Oils are shunted to the cleaning plant to remove solids. The oil-cleaning plant is located conveniently to both the old and new plants and handles oils for both. It is housed in a dustproof and fireproof steel and concrete building of two floors. Heaters on the upper floor bring the oils to proper temperature, after which gross solids are removed in a slow-speed bulk centrifuge similar to the type used in sugar refineries. The finer solids are then removed in a battery of high-speed centrifuges, after which the clean oil is returned to the plant head tanks. Electrical control equipment is isolated from the centrifuge room and the air in the building is conditioned in order to minimize fire risk. It is hardly necessary to say that this oil-cleaning plant is a very profitable investment.

The new secondary plant as well as the old plant and the primary plant are serviced by a dust-collecting system (Fig. 8) and the collected dust is reclaimed in a reclamation plant. The ore as it is received from the mine carries but 1.5 per cent moisture and because of the soluble sulfates of copper contained cannot be moistened to lessen dusting. Although dusting is not excessive, there is sufficient to warrant its collection. The

dust plant consists of 11 blowers with a total capacity of 130,000 cu. ft. per minute, and a system of flues by which dust is picked up at the discharge chute of the machines and other vital points. The ducts are of steel incased in masonry, located underground, and lined with rubber at

TABLE 2.—*Conveyor-belt Data*

Conveyor No.	Size Ore Handled, In.	Width Belt, In.	Number of Plies	Length Center to Center of Head and Tail Pulleys	Horse-power	Speed, R.p.m.
1	9	54	10	262′	300	590
3	9	48	12	184′	100	565
17	9	60	12	400′ 10″	150–300	315–630
18	9	60	12	427′ 4″	150–300	250–500
19	9	60	12	342′ 6″	150–300	250–500
20	9	60	12	45′	40	510
4	9	36	10	300′	40	395
4-A	9	36	12	146′	30	354
4-B	9	40	12	13′	15	343
4-C	9	36	8	329′	40	329
5	9	36	10	243′	40	395
6	9	36	8	385′	40	423
7	3	36	8	195′	50	333
7-A	3	36	10	203′	40	333
8	3	36	12	167′	40	333
9	3	36	8	159′	50	377
10	Vat feed	36	8 and 12	161′	40	436
11	Vat feed	36	8	163′	50	500
12	Vat feed	36	8	163′	50	383
13	Vat feed	48	10	371′	120	452
14	Vat feed	60	12	1257′	300(2)	270/540
15	Vat feed	48	10	272′ 6″	150	362
16	Vat feed	48	12 and 14	1019′	175	457
22	3	36	8	181′ 6″	50	300 max.
23	3	36	8	181′ 6″	50	300 max.
24	3	36	8	181′ 6″	50	300 max.
26	Vat feed	48	10	347′ 6″	150	375
27	Vat feed	48	10	295′ 3″	150	375
28	Dust	18	6	38′	4	101
29	Dust	18	6	266′	4	101
Loading bridge No. 1	Vat feed	60	12	140′ 6″	75	359
Loading bridge No. 2	Vat feed	48	10	131′	75	538

necessary locations. With all fans running 130,000 cu. ft. per minute of dust-laden air is delivered to seven cyclones, where the coarse dust drops out and is discharged continuously from the cyclone to a screw conveyor. After leaving the cyclones the dust enters seven baffled dust chambers

with hopper bases equipped with seven screw conveyors to pick up the settled dust which joins that from the cyclones. The chambers discharge to a 16 by 16-ft. flue in which there is a five-unit Cottrell precipitator, where the finest of the dust is recovered. All dust is moistened in a pug mill and is discharged on to the ore conveyors from either of the two secondary plants. Total recovery of dust is approximately 99 per cent and the tonnage at full production is 250 metric tons per day of 4 per cent grade. The operation more than pays for itself and of course results in much better working conditions for men and equipment.

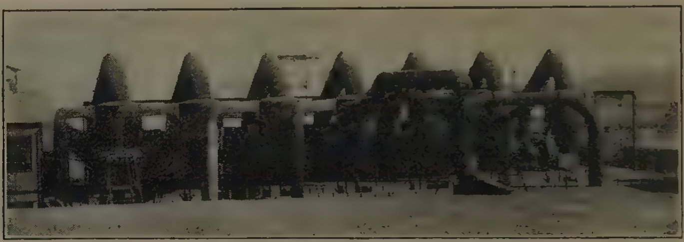


FIG. 8.—DUST-COLLECTING PLANT.

The crushed ore screens as follows:

THROUGH	ON	PER CENT
	0.371	10
0.371	0.093	56
0.093	0.0232	19
0.0232	0.0058	10
0.0058	0.0029	5
0.0029		2

This represents ore at the sample towers to which collected dust has been added. The screen characteristics are very well adapted to percolation leaching, the fines and oversize representing a comparatively small proportion of the ore, which ranges within the limits where best and fastest extraction is obtained. No doubt this excellent sizing is in part due to the character of the rock, although the type of crushers used is an important factor. It has been our experience that the centrifugal discharge machines have distinct advantages over the gravity discharge for the crushing of ore that is to be leached by percolation. For this reason we do not find cone crushers suitable in the final stages of crushing, although they have distinct advantages over the gyratory with conventional bowl and liner design.

All chutes in the plant are of sheet steel lined with manganese wearing plates, which are a decided improvement over wood chutes, especially with hard ore. The hardness of Chuquicamata ore may be gaged from the fact that 13.7 lb. of manganese steel is worn from disks per

1000 short tons of ore crushed. Almost all manganese steel wearing parts, both for the crushers and for the mine, are cast in our own shops.

All conveyors are inspected daily and defects are repaired with portable vulcanizing equipment. The dry climate and strong sunlight at Chuquicamata provide a severe test for conveyors that cannot be sheltered. A somewhat softer cover is required here than on indoor or shaded belts. Conveyor and transmission belting is stored in an underground air-conditioned room in the primary crushing plant, where stocks are carried with safety.

Tramp iron of small size is taken care of by the magnets, and large pieces, such as drill casing, are guarded against by payment of bonuses both at mine and plant for detection. The maintenance of crushing machinery is highly organized, and delays due to breakdowns are rare. One shift per day is used for repairs, the plant operating the other 16 hr. or as required.

FIG. 9.—LEACHING VATS, LOOKING EAST.
Solution sumps and crushing plant may be seen in background.

LEACHING PLANT

Equipment

The ore is leached in 13 leaching vats, each of approximately 11,500 short tons net capacity. These vats are arranged in two units, one containing seven vats placed to the east, and the other of six vats to the west of the crushing plants (Figs. 9 and 10). Each unit is served by one loading and two excavating bridges of the gantry type, the latter spanning the former. Two main loading conveyors run the length of the north side of the two vat units and four tailings trains operate on the south side, taking the discharge from the four excavating bridges. Each vat is 150 ft. long by 110 ft. wide by 16½ to 18½ ft. deep to the top of the filter

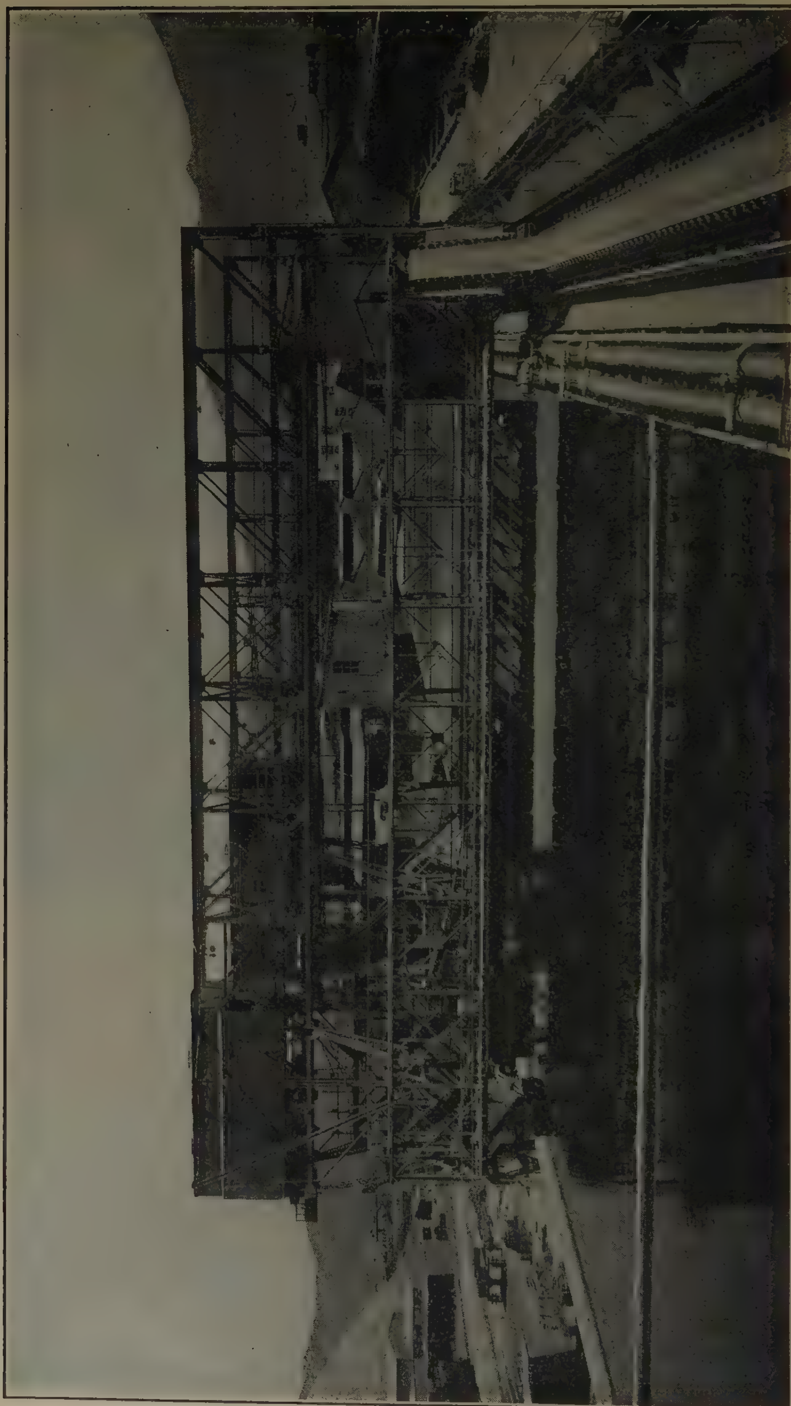


Fig. 10.—West leaching vats.
Vat-loading bridge in foreground and behind it one of the unloading bridges.

bottom. Vats are constructed in blocks of three or four of reinforced concrete and are set on piers to facilitate inspection of the vat bottom. A mastic sand mixture is used to line the vats for protection from acid. This mastic is reinforced with expanded metal tied to the side walls and bottoms. The linings are 4 in. thick and are cast in place on the walls, using collapsible metal forms. Mastic linings are cheap and effective. Repairs are easily effected and worn or broken linings can be completely recovered by remelting. Protection from sun is essential to long life, however.

The vats are equipped with filter bottoms constructed as follows: A false bottom is laid on the mastic floor of the vat, composed of 6 by 6-in. pine 6 ft. long laid end to end 18 in. apart, the ends being beveled to allow passage of solution. Across these are laid 2 by 6-in. pine planks set ¼ in. apart. In this false bottom are left eight spaces 10 ft. square for draining the vat. Into these spaces are inserted pre-built filter units composed of 4 by 6-in. timbers set 12 in. apart, across which are laid 2 by 6-in. planks 3 in. apart. Over this is laid coco matting, and on the matting in reverse direction to the lower planks are laid more 2 by 6-in. planks ¾ in. apart. This bottom will drain up to 1500 cubic meters per hour through average ore, and is cleaner and more accessible for repair than the conventional type of filter bottom in which the entire vat bottom is of filter construction. Leaching records show no difference in extraction from vats equipped with the two types of bottoms. The filters are laid down with the top layer of 2 by 6-in. timber parallel to the closing movement of the excavating buckets. In the excavating operation a small tonnage of tailings is customarily left on the filter, and these two practices greatly extend filter-bottom life, so that but with occasional minor repairs it is usual for a bottom to last eight years. The filters are kept clean by running the first covering solution for each ore charge on to the vat through the bottom.

Twenty-five solution sumps placed to the north of and parallel to the ore vats and four auxiliary storage sumps located beyond the dechloridizing plant serve the leaching vats. The total capacity of these sumps is 75,000 cu. m. They are placed on higher ground than the vats and solutions are run from them by gravity. Their construction is similar but somewhat lighter than that of the leaching vats. The sumps are used for transient storage of treatment advance and wash solutions and as buffer tanks for spent and strong electrolyte.

The sump capacity may seem excessive but it is a necessary feature of the batch type of leaching, and together with the vats provides a tremendous copper-storage capacity for emergencies. The large solution-storage capacity serves also to suppress variations in acid and impurity content of the electrolyte even over extended periods of variation in the ore in respect to these two.

Volume measurements in the sumps are taken by means of indicators composed of a counterweighted float protected from surge. The acid-proof line from the float to the counterweight carries an arrow and passes over a horizontal frame on which is bolted a copper strip 12 ft. long graduated in cubic meters and in feet. This indicator is converted to a recorder by using a milled soft iron counterweight, the vertical movement of which is reduced in reference to that of the float by pulleys, and which travels in a solenoid. The variations in induction are calibrated in feet of depth, or cubic meters, and are recorded on a circular chart.

When solution is run from sumps to the top of a vat, displacing solution withdrawn from the bottom and pumped to other sumps, the rates of flow are synchronized by an adjustable level marker fixed on the vat wall and protected from surge and wind. Throughout the operation the level of solution in the vat is maintained the same with regard to the fixed position of the marker.

Solutions are carried throughout the plant in wood-stave pipe lines. These lines are of the discontinuous type and are made up in 17-ft. lengths. Sections are coupled with wood-stave couplers and many sections terminate in cast-iron bell flanges to which the cast-iron fittings are bolted. Pipe is milled and made in our shops from selected Oregon pine and Douglas fir. The staves are assembled with mastic joints and steel hoop bolts with malleable-iron shoes. Pipe fittings are of cast iron mastic-lined, utilizing collapsible forms, although today we are gradually changing to hard lead linings, similarly placed in the cast-iron shell. Four sizes of pipe are in use; 8-in., 12-in., 15-in. and 24-in., depending upon the service. There are in the leaching plant and tank house pipe-line system 68,865 ft. of pipe with 885 valves and 2090 elbows and tees. Wood-stave pipe is the best material we know of for pressure lines required to resist the attack, when air is not present, of a sulfuric acid electrolyte containing ferric iron, chloride and over 1.0 per cent of nitric acid. The success of wood-stave pipe in acid service is based on the prevention of leaks which subsequently concentrate in the sun and char the wood from the outside. We find practically no corrosion in the interior of pipes that have failed thoroughly on the outside. The prevention of leaks is largely concerned with accurate turning of butts and couplers, accuracy in fitting flanges and careful assembly in the field. Every line is pitched to drain at one or two points and the end flanges are fitted with a 2-in. vent valve, which is opened to prevent collapse before a line is drained for repairs. Four main groups of solution lines run parallel to the vats and sumps, one on the lower, another on the upper side of the vats, and two more on the lower side of the sumps. These lines are connected to two pump houses, one to the west of the first ore vat and the other between the sixth and seventh vats south of the sample towers. The gravity lines are connected with four crossover lines passing between the two blocks

of the vat units on both the east and west side. The installed pipe is in excess of actual requirements in order to obtain flexibility and to facilitate repairs. The average life of wood-stave pipe in acid service in pressure lines is 5½ years.

Solutions are handled by nineteen 15-in. and four 9-in. vertical centrifugal pumps with hard lead-covered runners, casings and boots. The intake of these pumps is connected to mastic-lined reinforced concrete sumps equipped with cast-lead screens. These sumps act as traps for tailings that may be carried through the filter bottom and down the pipe lines if the filter is broken by the excavating bucket. This type of pump is reliable but its efficiency is surpassed, of course, by the horizontal centrifugal pump. As high as 250,000 cu. m. is pumped daily against a 70-ft. equivalent head, and operating delays due to pump failure are very rare.

Automatic sampling devices are attached to the pump sumps and every solution produced from the vats is sampled by these devices. A relatively large sample is taken, cut down and deposited in a cool, covered wooden tank. The samples are extremely accurate, because the pumping rate is uniform.

Operation

The method of leaching may be termed the batch percolation system in distinction to the counter-current percolation system in use elsewhere. Extraction is superior with the counter-current system but the batch system is the more flexible of the two.

The ore is loaded either into solution or into a dry vat, depending upon the relation of the time of ore loading to the time of production of the treatment solution coming from another vat. We do not find any difference in extraction between loading into solution or into a dry vat. The ore is not bedded but is loaded in one operation from the bottom to the top before the bridge is advanced. Deposition on the filter of windblown fines from the ore stream is avoided by advancing the loading bridge into the prevailing wind. Curtains suspended from the bridge shield the stream from the wind as much as possible. After loading the ore is leveled by hand and the fines on the surface of the ore are turned in. Slime pockets are avoided by this simple operation. After leveling is finished the remainder of the treatment or covering solution is run on through the bottom and the ore is submerged to the marker level. Approximately 3500 cu. m. of solution is required to submerge 10,000 metric tons of average ore containing 1.5 per cent of moisture. Of this volume 750 cu. m. is absorbed and 2750 cu. m. is drainable.

After submergence the ore is allowed to soak in the treatment solution from 8 to 24 hr., then the production of first strong solution is started with a solution called first advance going on the top. This solution is

produced to a cutoff limit and its volume varies from 3000 to 6000 cu. m., depending upon the ore grade. Following the production of the first strong solution, a second soaking period ensues, which varies from 24 to 72 hr., after which the production of second strong solution is started with a solution called second advance going on top. This second strong solution is likewise produced to a cutoff limit and its volume varies within practically the same limits as that of the first strong solution. A third soak of variable duration then follows, after which first advance solution is produced with the volume advance and spent electrolyte going on top. This first advance is produced to a volume somewhat less than the volume of the first strong previously produced. A fourth soak follows when second advance is produced with spent electrolyte going on top. The fifth and final soak then follows, after which the washing process starts. Six washes of decreasing copper grade from 9 to 1 grams per liter followed by fresh water are put on top of the vat successively and treatment or covering solution, volume advance the six washes and volume discard are produced in that order. The vat is then drained for at least 4 hr., after which it is ready for excavation.

The amount of the volume advance solution produced is determined by the acid gained from the ore and is adjusted from time to time to suit. This volume is equal to the volume of primary spent electrolyte stripped and wasted plus the volume of evaporation taking place throughout the primary solution system. The volume discard solution is produced at the end of washing and is made to increase the ore-washing efficiency and to provide a slightly acid solution for washing of cement copper to be used in dechloridizing.

The first strong solution from one vat and the second strong solution from another vat are produced simultaneously. The strongs in this way are thoroughly blended, resulting in a uniform grade for the tank house. The operation is a compact one and has proved out well in practice. The inclusion of two advance solutions in the leaching process represents a concession to the practice employed in the efficient counter-current system of leaching. Leaching operations are scheduled on a time chart days in advance. This chart system of control prevents conflicts and gives the supervision a very close grasp of operations. Ninety-six hours is the minimum cycle for good operation. The cycle increases, of course, with lower ore tonnage. The plant is capable of handling a maximum of 1,400,000 short tons of ore per month and is thus quite well synchronized with crushing capacity.

The oxide ore is composed of a highly altered granite containing veinlets of as well as disseminated brochantite and other minor oxide minerals and a small amount of sulfide. The ore crushes cleanly and percolates freely, being admirably suited to percolation leaching. The mill product runs 90 per cent through 371 screen and contains but 2 per cent of fines

that will pass 100 mesh. The mineral bronchantite supplies approximately 0.40 kilos of sulfuric acid per kilo of copper leached from the ore, which is sufficient acid to operate the plant with no other acid source required. The ore is noted for its astonishingly low impurity content, as will be shown in the typical analysis of today's ore and the values recovered, shown in Table 3.

TABLE 3.—*Ore Values and Recovery*

	Per Cent	Kilos per Metric Ton Ore	Kilos Metric Ton Ore Extracted	Extraction, Per Cent
Copper	2.100	21.0	19.5	93.0
Iron	0.95	9.5	0.15	15.8
Chlorine	0.012	0.12	0.04	33.3
Arsenic	0.002	0.02	0.01	50.0
Antimony	0.001	0.01	0.004	40.0
Molybdenum	0.010	0.10	0.01	10.0
Nitric acid	0.015	0.15	0.08	53.3
Sodium	1.10			
Potassium	5.20			
Silica	68.0			
Al_2O_3	16.0			

The present ore contains approximately 90 per cent of oxide copper and 10 per cent of sulfide copper. Of the former, 98 per cent, and of the latter, 40 per cent is extracted in the leaching process. Ore containing 60 per cent of oxide and 40 per cent of sulfide, called border-line mixed ore, is also leached today in moderate tonnages. This ore is treated with large volumes of solution containing ferric iron. Chemically equivalent tonnages of scrap iron are saved in the dechloridizing operation by the ferric iron reduced by sulfides in the ore. The sulfides in the ore are largely chalcocite and pyrite but very little of the latter is decomposed. As high as 70 per cent of the sulfide copper is extracted from mixed ore.

The spent electrolyte as it goes on the ore carries 2.0 grams per liter of ferric iron and its temperature is 30° C. The strong solution produced carries 0.8 grams per liter ferric iron and its temperature is 20° C. The electrolyte is therefore heated approximately 10° C. owing to its resistance in electrodeposition. These temperatures are not conducive to the best results in the leaching of sulfides. No heating of electrolyte other than that caused by its resistance in electrolysis is resorted to. At a high production rate the leaching period is shortened but the electrolyte is 10° to 15° C. hotter and leaching results are about equal both for oxides and sulfides.

Table 4 shows the composition of spent electrolyte of 1928 and of 1933. The lower impurities now present are due to higher acid gain from the ore

plus decreased impurities contained in the ore coupled with lower extraction of impurities due to lower free acid. Furthermore, the ore of today is harder than heretofore and the gangue is not decomposed to the same extent as formerly. This feature is also partly responsible for increased acid gain, which is illustrated by the fact that an ore of ⅜-in. sizing which shows 8 kilos acid gain per metric ton will show an acid loss if pulped and leached. Many of the basic compounds in Chuquicamata ore are not copper bearing and if a large percentage of fines were present their separate treatment would be desirable from this standpoint alone.

TABLE 4.—*Spent Electrolyte, Grams per Liter*

	1928	May, 1933
Copper	14.9	15.6
H_2SO_4	70.0	60.0
HNO_3	3.51	1.50
Total iron	4.50	2.40
Ferric iron	1.30	1.90
Ferrous iron	3.20	0.50
Molybdenum	0.19	0.30 to 1.00
Arsenic	0.31	0.22
Antimony	0.02	0.01
Selenium	Tr.	Tr.
Tellurium	None.	None
Calcium	0.50	0.40
Sodium	7.82	6.20
Potassium	2.31	1.75
Aluminum	3.51	2.10
Magnesium	1.33	0.75
Chlorine	0.18	0.12
Total solids	207	155
Degrees Centigrade	40.0	31.0

Copper extraction has not altered with the lower free acid and the lower spent electrolyte temperature is practically offset by longer leaching cycle. While the electrolyte was more active in 1928 owing to oxidizing effect of higher nitric acid, this is offset by higher ferric iron in the spent electrolyte of today because of the oxidation of ferrous iron by lead-antimony anodes, which is greatly superior to that by Chilex anodes. Increasing sulfides in the ore and stable electrolyte has prevented an increase of load on the dechloridizing plant through higher ferric iron in the spent electrolyte.

Slag, magnesite and silica cobbings from the refining furnaces are leached in an ore-leaching vat after crushing and removal of the metallics in a jig. In order that these products may be leached by percolation, they are milled to ⅜-in. screen only. Metallics recovered are equal

approximately to 33 per cent of the total fine copper content of these secondaries, and the coarse concentrate runs 90 per cent of copper. This is then crushed in ball mill and reconcentrated to 99 per cent of copper, which is composed of pure metallics carrying about 1.0 per cent of slag. The hutch product from the jig is similarly reconcentrated. All

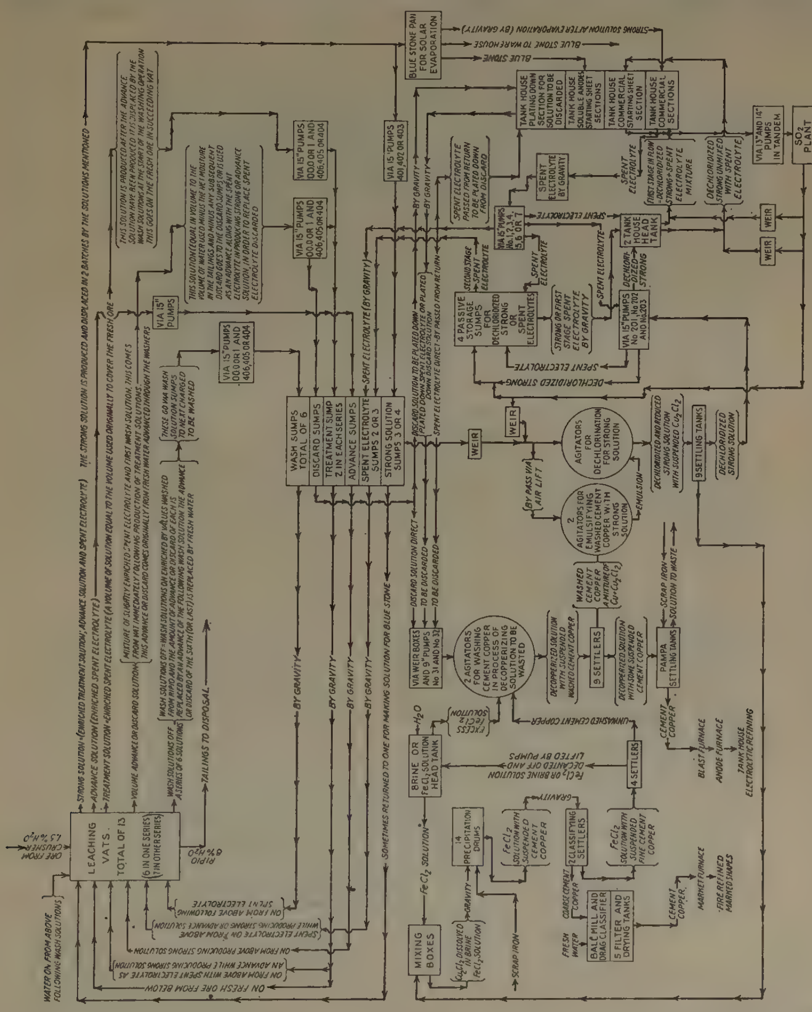


FIG. 11.—FLOW SHEET OF LEACHING AND DECHLORIDIZING PLANTS, 1933.

concentrates are returned to the reverberatory melting furnaces and the jig tailings are loaded into a leaching vat. The material to leaching carries on an average 25 per cent of copper as oxide and silicate but decomposition of the latter is slow, requiring about 12 months. However, as the tonnage is small in comparison to leaching capacity, this time is of little consequence.

This leaching is carried on with electrolyte about to be stripped and discarded, which prevents re-entry of impurities into the solution system. The leaching extraction is about 85 per cent and the over-all recovery from the original secondaries is 90 per cent. The slag shows no tendency to gelatinize although fines increase with the leaching time. The percolation rate however remains ample.

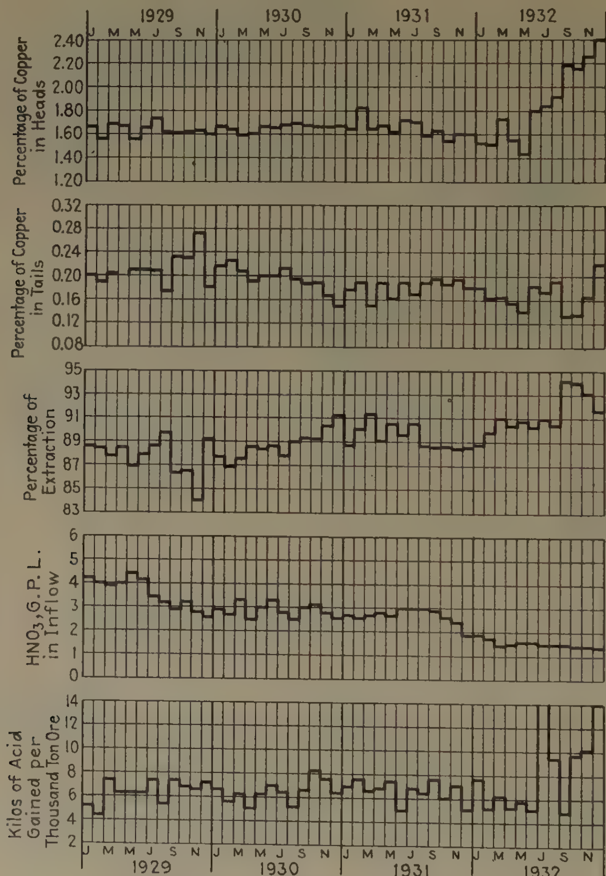


FIG. 12.—LEACHING DATA.

Fig. 11 shows the flow sheet; Figs. 12 and 13 and Tables 5 and 6 give leaching data.

Tailings Disposal

Each bank of vats is served by two gantry tailings-discharging bridges. These bridges are of three types, the most modern embodying dynamic breaking and Ward-Leonard control. Buckets are of the clamshell type and vary in capacity from 8 to 12 tons. Two bridges operate

together on a single vat discharging an average of 11,000 short tons of tailings in 7 to 7½ hours.

Bucket lips are tipped with manganese steel and as ore washing is thorough there is little corrosion. The closing movement of the bucket is in the direction of the upper layer of timber of the filter bottom, which successfully prevents nipping of the filter bottom, an occurrence that is

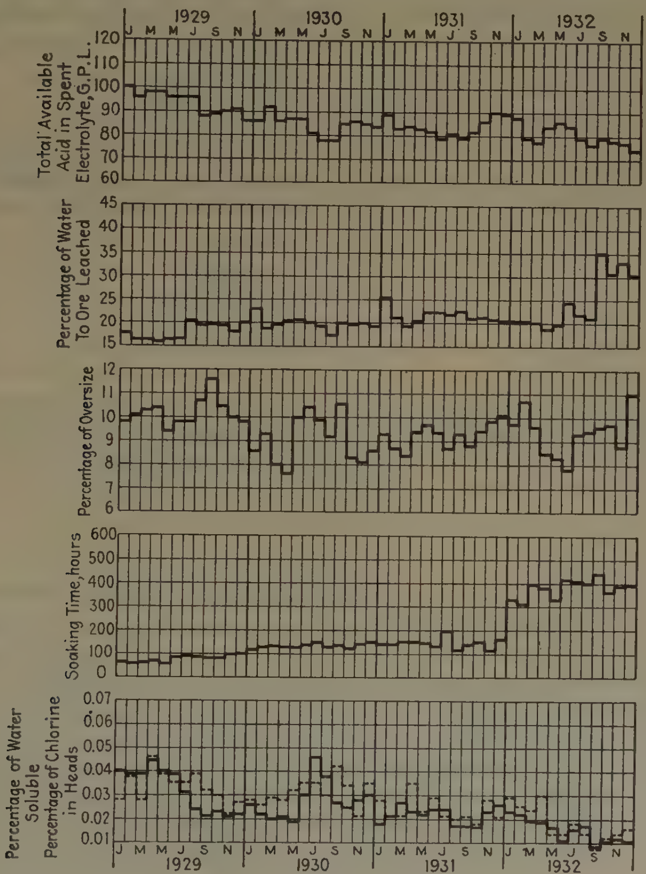


FIG. 13.—LEACHING DATA.

detrimental to both the bucket and the bottom. The vats are not dug clean, there being approximately 800 tons of tailings left on the bottom. This practice protects the bottom and the bucket and increases the digging rate. It also weights down the bottom, preventing it from floating if a covering solution is run on before the ore is loaded.

The buckets discharge into hoppers built in the pier leg from which the tailings are run into 12-yd. dump cars in trains of 26, each handled by a 70-ton electric locomotive. The fan-shaped tailings dump is 4.5 km.

in circumference and ranges from 25 to 75 m. in height. Power is taken from third rail in the yards and on the permanent portion of the dump. The active track on the dump is equipped with a flexible trolley system, which is mounted on the ties and follows the changing contour of the dump as the track is shifted. Increasing trolley length is paid out from a reel at either end of the system, the counterweighted trolley passing freely through the flexible hangers. This installation operates with perfect success.

Bridges are anchored to the rails when not actually in use and although high winds are encountered at times there has never been any accident from that cause at Chuquicamata. Careful attention is paid to equalizing the bridge-drive mechanism.

Trains from the east vats enter the east side of the dump and those from the west vats enter the west side, making the average haul per round trip approximately 2.25 km. at the present time. A round trip requires 43 minutes for a net 550-ton train, including loading and discharging. The power consumption equals 0.28 kw-hr. per ton for the excavating bridges and 0.04 kw-hr. per ton for the locomotives.

Data on tailings disposal are given in Table 7.

TABLE 5.—*Solution Data—Leaching*[a]

	First Strong Solution	Second Strong Solution	1st Ad.	2d Ad.	Covering or Treatment Solution	Spent Electrolyte	1st Wash	2d Wash	3d Wash	4th Wash	5th Wash	Volume Discard	6th Wash	Water
Volume	5000	5000	3500	3500	3500	8800	3000	2000	2000	2000	2000	300	3000	2350
Degrees Centigrade	15°	22°												
Copper, grams per liter	42.0	32.0	22.0	19.0	17.0	15.0	9.0	7.0	5.5	4.0	2.5	1.5	1.0	
H_2SO_4	25.0	32.0				58.0								
Iron	2.5	2.5				2.5								
Ferric iron	1.7	1.0				2.0								
Ferrous iron	1.8	1.5				0.5								
Nitric acid	1.5	1.5				1.5								

[a] Approximately 100 cm. per vat average evaporation.

Dechloridizing Plant

The dechloridizing plant (Fig. 14) has five functions:

1. Precipitation of chlorine as cuprous chloride from strong solution.

2. Reduction of ferric iron in strong solution.

3. Stripping of copper from electrolyte to be wasted.

4. Recovery of cement copper from cuprous chloride by solution in $FeCl_2$ and cementation on scrap iron.

5. Preparation of a high-grade cement copper suitable for melting in the reverberatory furnace for the production of a high-grade fire-refined copper.

Figs. 15 and 16 show sections of the dechloridizing plant. Strong solution from the leaching-plant sumps enters the plant by gravity

TABLE 6.—*Leaching Plant Data*

Leaching-plant capacity, short tons per month	1,400,000
Short tons per leaching charge	11,500
Cycle time, days	4–16
Extraction percentage	88–94
Acid gained, kilos per metric ton ore	6–12
Copper recovered, kilos per metric ton ore	14–20
Iron dissolved, kilos per metric ton ore	0.15
Molybdenum dissolved, kilos per metric ton ore	0.01–0.05
Chlorine dissolved, kilos per metric ton ore	0.04
Moisture in tailings, per cent	9.0
Water per metric ton ore, liters	235
Total available acid in electrolyte, grams per liter	80–90
Temperature of leaching solutions, deg. C	14–30
Copper in spent electrolyte, grams per liter	14–16
Copper in strong solution, grams per liter	32–45
Copper in first wash, grams per liter	9.0
Copper in sixth wash, grams per liter	1.0
Water-soluble content of tailings, per cent	0.01–0.02

FIG. 14.—REAR OF DECHLORIDIZING PLANT. TYPICAL WOOD-STAVE PIPE LINES IN FOREGROUND AND ROTARY MASTIC MIXERS AT LEFT.

TABLE 7.—*Tailings Disposal Data*

Capacity (4 excavating bridges), short tons per hour	3150
Tailings per train, short tons	550
Kilometers per trip	2.25
Cycle time train trip, minutes	43

through three 15-in. wood-stave lines. Of this solution 10 per cent enters one or more Devereux mechanical agitators equipped with bronze shaft and impeller and driven through bevel gear by 50-hp. motor. Into these agitators is also charged cement copper elevated by crane bucket and fed through hoppers. The resulting pulp leaves the agitators through top and bottom discharges and is dropped through pipe lines into the remaining 90 per cent of the strong solution passing in an adjacent launder. The mixture then enters one or more Parral agitators, where the ferric iron is reduced and the chlorine is precipitated as cuprous chloride. The Parral agitators, of which there are three, which may be run in series or parallel, discharge at the top into launders that convey the solution and suspended cement and cuprous chloride to nine settling tanks, each 38 by 28 ft. by 12 ft. deep. Here the solids drop out and the reduced and dechlorized strong solution passes through launders to a pump house, where it is elevated to the head tanks of the electrolytic tank house.

These cuprous chloride settlers are cut out of service in rotation and drained by pumps, and after the contained cuprous chloride is washed with water it is dug and elevated by clamshell bucket to small mechanical agitators where it is treated with hot ferrous chloride brine carrying 100 grams per liter of chlorine. The cuprous chloride dissolves and the mixture of solution and cement is run into seven sets of rotating bronze drums charged with scrap iron, each set being composed of two drums in tandem. The scrap iron is charged through a removable door from a charging floor above the drums. Scrap iron is cut up with arc torches and moved to this floor over the industrial railroad.

The copper in solution is cemented in these drums and passes to settling tanks. The brine solution runs out of the settlers to a sump tank and from there to pumps by which it is elevated to the cuprous chloride-dissolving agitators, a small volume being bypassed to the stripping agitators.

The coarsest of the cement is excavated, pulped with water and passed through a ball mill and Dorr drag classifier. The fines are returned to process and the coarse passes to filter bottom tanks, from which after further washing and draining it is subsequently dug and shipped to the smelter, where it is dried, melted in a reverberatory furnace and after refining is cast into fire-refined shapes. This cement is similar in sizing to sea sand and weighs 110 kilos per cubic foot.

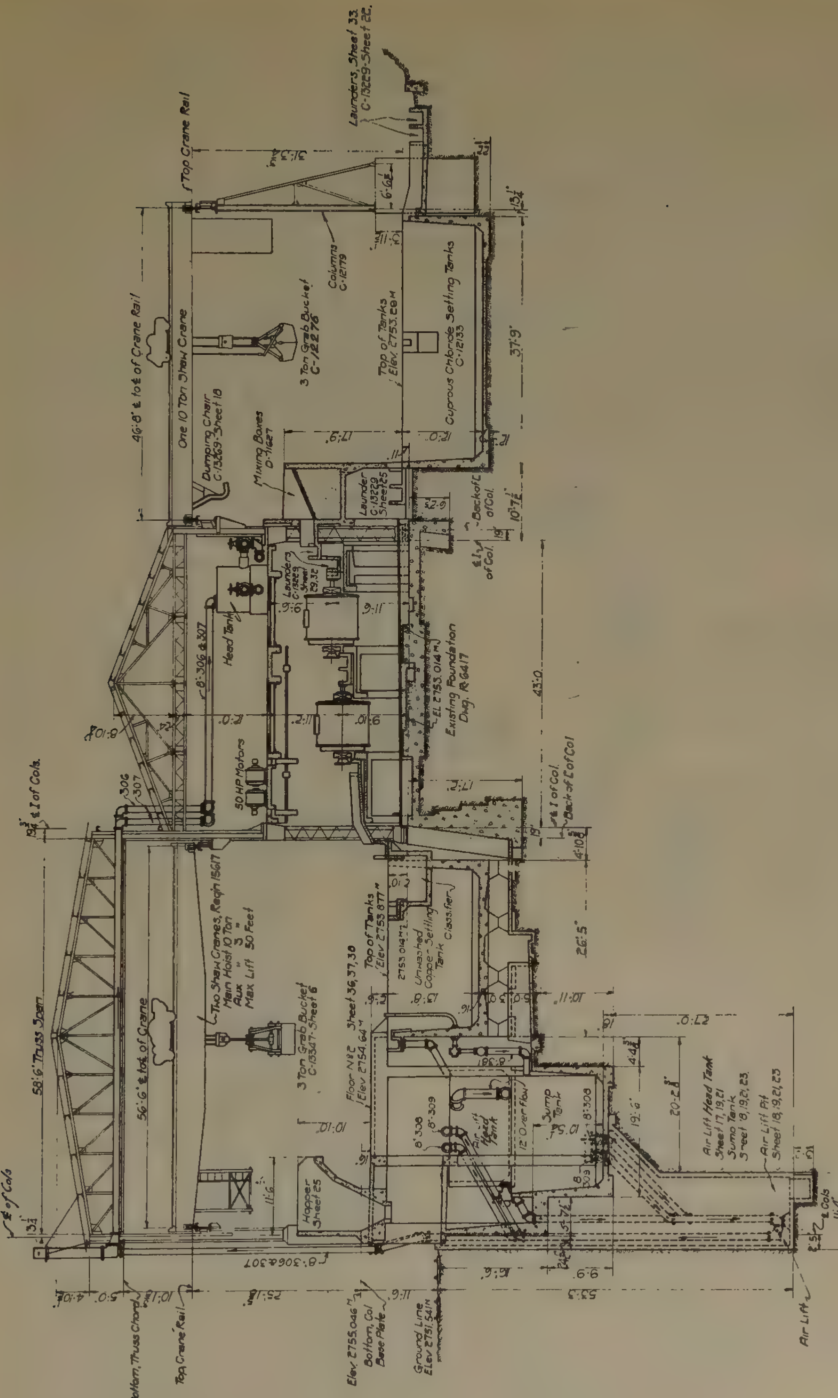


FIG. 15.—SECTION THROUGH WEST END OF DECHLORIDIZING BUILDING.

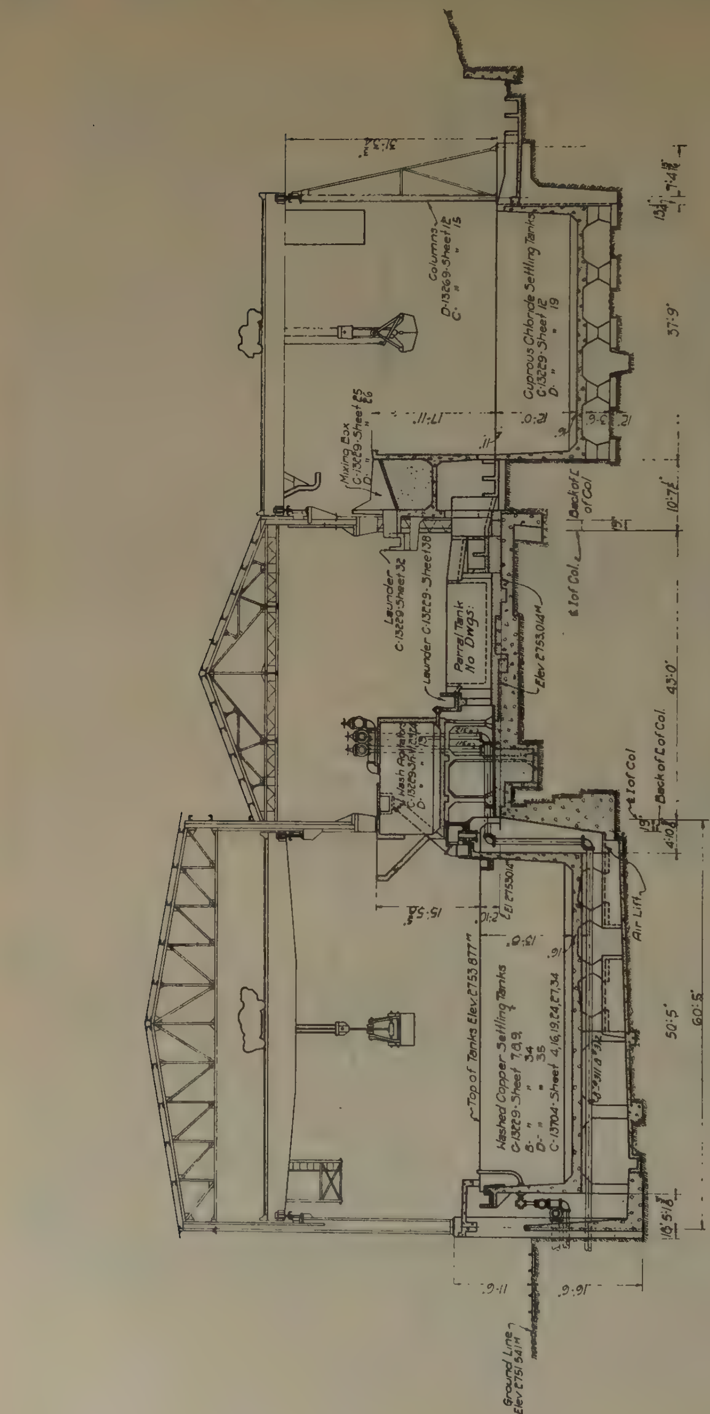


FIG. 16.—SECTION THROUGH EAST END OF DECHLORIDIZING BUILDING.

The remainder of the drum cement is dug and fed by crane bucket to a Devereux agitator, into which is run a small amount of $FeCl_2$ brine, and the electrolyte that is to be stripped and wasted. Copper is precipitated from the electrolyte as cuprous chloride. The pulp passes to settlers and the barren solution overflowing the settlers is piped to the pampa, where it is wasted. In this operation a small excess of ferrous chloride and, as in the dechloridizing reaction, a large excess of cement, is required to force the reaction in the limited time available.

The solids in the settlers carry about 7 per cent of chlorine. This product is dug and washed in two Devereux agitators in two stages. In the first stage the washing is accomplished by the electrolyte prior to entering the stripping agitator, and in the second stage by the leaching-plant discard solution made before or after sixth wash is produced. The pulp passes to settlers after each wash and the runoff is returned to the stripping agitator.

The final washed cement is then ready for use in dechloridizing. It is dug by crane and elevated to a Devereux agitator, where it is pulped with strong solution passing then to the dechloridizing Parrals. This completes the cycle.

Four traveling cranes handle the cement through the various operations and at full production move 2500 wet tons per day. Settlers and mechanical agitators are built of reinforced concrete construction, mastic lined, and Parrals are of cast mastic reinforced. Clamshell buckets, brine pumps, agitator shafts and paddles are of bronze.

The precipitation of chlorine as cuprous chlorine takes place readily and goes practically to completion in a few minutes when excess of copper of the proper fineness is present. The solution of cuprous chloride in ferrous chloride brine requires a rich hot brine. The dissolving action at best is slow, but it is essential to the process that as near complete solution as possible be obtained before cementation, otherwise cuprous chloride will be cemented in place on the copper particle, causing undesirable grain growth. The control of grain size of the copper cement particles is necessary to the successful operation of the plant. If too coarse the material reacts slowly, cannot be dug by crane, and plugs launders. If too fine, cuprous chloride fails to settle completely and is carried to the tank house and to the pampa. The grain-size control is located in the cementation operation and requires proper brine gravity and temperature. Copper of the proper grain size weighs approximately 50 kilos per cubic foot.

The iron consumed amounts to 50 kilos gross per 100 kilos of copper precipitated from cuprous chloride. This does not take into consideration ferric iron that is reduced in discard solution nor the impurities in the scrap iron. Brine solution, which was formerly handled by air-lift, is now pumped to cut down the oxidation of ferrous chloride.

The amount of chlorine precipitated from strong solution has gradually diminished as the surface ores have been depleted, until today it is insufficient to operate the plant. Sodium chloride crystallized from desert springs a few kilometers from the plant are added to balance the operation. Approximately 45 per cent of the required chlorine is derived from the ore and 25 per cent from the water used in washing the ore, and 30 per cent is added in the form of NaCl.

Precipitation as cuprous chloride is probably the most efficient and satisfactory method yet devised for stripping solutions completely of their copper. This is especially true at Chuquicamata, as a large part of the cement so derived is subsequently dissolved by ferric iron in strong solution and the electrodeposition efficiency is thereby increased. The net result is equivalent to complete electrostripping at an efficiency comparable to that of normal cathode production with high-grade solutions, and of course results in hard dense cathodes of high purity. Despite the fact, therefore, that a time may conceivably come when dechloridization of electrolyte is no longer necessary, it will undoubtedly prove economical and desirable to add cheap sodium chloride to the ore in order to retain the advantageous features that this plant contributes to operations.

The triple washing of cement prior to use in dechloridizing is made for the purpose of washing out soluble iron salts, thus minimizing the iron picked up by electrolyte in the dechloridizing plant.

TABLE 8.—*Dechloridizing Plant Data*

Dechloridizing capacity, cubic meters strong solution per 24 hr. .	25,000
Tonnage capacity to cementation, short tons per 24 hr.	140
Normal chlorine in entering strong solution, grams per liter. . . .	0.50
Normal chlorine in leaving strong solution, grams per liter.	0.05
Scrap-iron consumption per unit of cuprous copper.	0.50
Grade of solution to stripping, grams per liter copper.	8–40
Grade of solution from stripping, grams per liter copper.	0.4

The purity of the cement copper shipped for fire refining is notable. An equivalent tonnage of cement formerly joined the plant secondary products and was smelted in the blast furnace. The resulting black copper was refined in a reverberatory furnace and cast into soluble anodes, these anodes being used to manufacture starting sheets. This operation was thus ordered, since the production of starting sheets strictly from commercial electrolyte using insoluble anodes was not feasible because of the character of the electrolyte, which at times was not stable, a matter that will be discussed further on. However, with proper stabilization of the electrolyte, we now manufacture all starting sheets required from commercial solution and the sheets are of a quality superior to those made from soluble anodes and electrolyte derived from bluestone. The cement copper that was formerly used for soluble anode production is

therefore diverted now in equivalent tonnage although of different character to the production of fire-refined copper, and the smelter secondaries are jigged and leached as previously described.

Approximately 3000 tons of fine copper cement is carried as working stock in the dechloridizing plant. Up to 25,000 cubic meters of solution can be reduced and dechloridized per day. Various data are given in Table 8.

Sulfur Dioxide Plant

In 1930 a sulfur-burning and SO_2 absorption plant was constructed to stabilize electrolyte (Fig. 17). This was found necessary in order to prevent decomposition of nitric acid catalyzed by molybdenum salts and to dissipate nitric acid from solutions. Operating difficulties were encountered in the tank house and dechloridizing plant when ore containing over 0.008 per cent of molybdenum was treated. This trouble took the form of decomposition of nitric acid and oxidation of ferrous iron and of cuprous chloride. Once started, this reaction became self-catalyzed by oxides of nitrogen and spread rapidly throughout the entire solution system of approximately 100,000 cubic meters.

The solutions fumed strongly, giving off great volumes of oxides of nitrogen, and in this condition it was impossible to properly dechloridize or to strip waste solutions. Chlorine values pyramided in electrolyte, and cathodes carried prohibitive quantities of cuprous chloride. The active oxidation going on in solution corroded cathodes to a degree that decreased current efficiency to 60 per cent from a normal of 90 per cent. Anodes and fittings were also badly corroded. This active electrolyte attacked the ore vigorously and, while extraction of sulfides was improved, much larger quantities of impurities were picked up and the rate of acid gain from the ore fell off.

Experimental work developed the stabilizing effect of sulfurous acid in solution. Small quantities inhibited the decomposition of nitric acid in the manner described above and gradually dissipated nitric acid from solution without oxidation occurring. Test work developed simultaneously the feasibility of applying the limerock purification process that had been placed in successful operation at the Potrerillos plant of the Andes Copper Mining Co. By this process iron, molybdenum, arsenic and antimony would be precipitated and removed from solution, leaving an extremely pure electrolyte with consequent high power yield in electrodeposition. However, this advantage was offset by adverse extraction of sulfides contained in the oxide ore in the leaching operation owing to the decreased tonnage of ferric iron available. As there also was a substantial difference in costs of construction of the two plants and in the simplicity of operation, the sulfur dioxide treatment was decided upon.

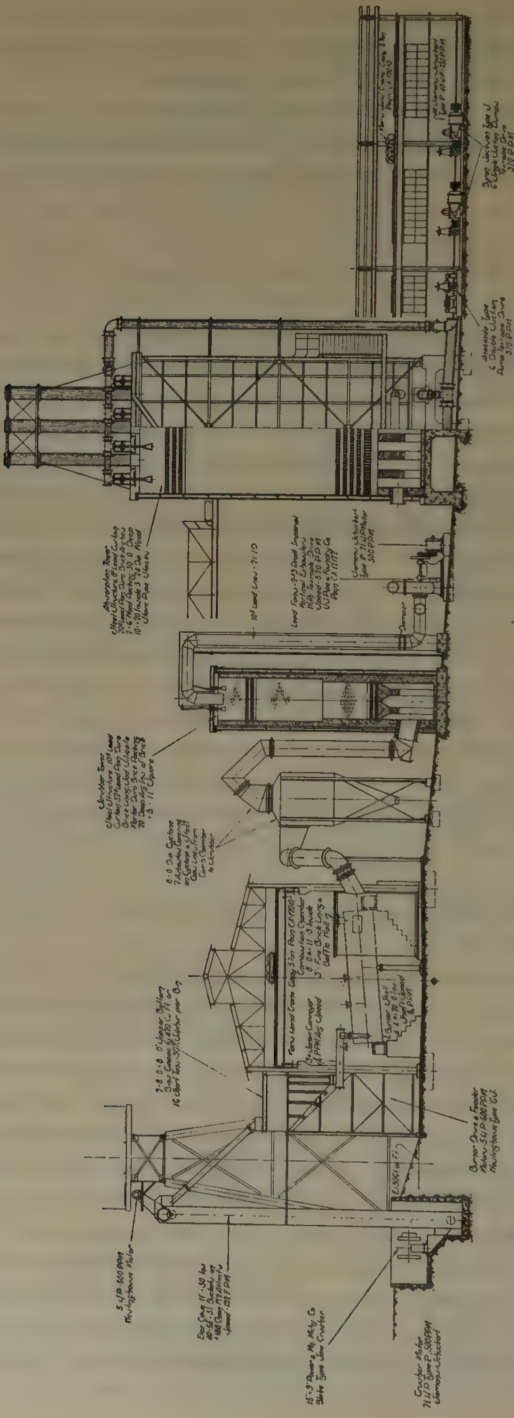


FIG. 17.—ELEVATION OF SULFUR DIOXIDE PLANT.

Operation at the sulfur dioxide plant consists of crushing and burning local volcanic sulfur ore and absorption of the gas in electrolyte, followed by distribution of the gassed electrolyte to necessary points in the solution system. The flow sheet is shown in Fig. 18. The sulfur ore is crushed in a Blake-type jaw crusher to ½-in. sizing and is elevated by bucket elevator to the bins, being sampled automatically as it enters the latter. Fires are rare with this layout, despite the fact that a rock gangue accompanies some native sulfurs. The type of burner in use is one adapted from a design by Dupont engineers and in use at their explosives plant at Rio Loa, some 30 km. from Chuquicamata. It is very well suited to burning native ores of from 45 to 90 per cent sulfur content.

The burner consists of a four-section plate-steel shell 20 ft. long and 4 ft. 4 in. in diameter. This shell may be lined with refractory brick for burning low-grade ores. A water-cooled shaft passes longitudinally through the shell and carries 30 rabble arms, the working face of which is set at 26.5° from the shaft and clear the inside of the shell by ¾ in. This shaft is rotated at 1 to 2 r.p.m. The burner is set at an angle of 25° from the horizontal, and sulfur ore is fed through the upper shell at the lower end by means of a screw conveyor driven through a Reeves speed reducer. The screw receives the crushed sulfur ore from an elevated bin of 15 tons capacity.

The sulfur is advanced through the burner by the rabbles, and at the upper end the ash drops into a chute and is delivered to a closed ash hopper. The gas and sublimed sulfur pass to a cylindrical refractory-lined and baffled combustion chamber, where burning is completed. The burner and combustion chamber are equipped with suitable air intakes and are placed under from 0.1 to 0.5 in. of vacuum by exhaust fans, which draw the gas through the burners, cyclone and scrubber tower and force the clean and washed gas into the absorption tower. There are two burners, bins, feeders and combustion chambers housed in the burner building.

The gas of 8 to 14 per cent grade, depending on the sulfur ore grade, passes to a cyclone, then to a lead-curtained, brick-walled and packed scrubber tower 5 ft. 6 in. square inside dimension by 32 ft. 8 in. high. Two lead-cased fans, one of which is a spare, with lead-covered steel rotor, are between the scrubber tower and the absorption tower. The latter, which is a lead-curtained tower with lead pan, measures 10 by 20 ft. by 44 ft. high, and is packed with 2 by 4 pine lumber. The two towers are served by 3 by 4 in. horizontal lead centrifugal pumps, one of which is a spare, at the tower level in closed circuit with a 15-in. vertical centrifugal pump at pump house No. 1 east of the tank house, which picks up the outflow from the starting-sheet sections.

Eight spray nozzles of the type used in acid towers distribute 300 to 400 cu. m. per hour in this tower, while four similar but smaller sprays

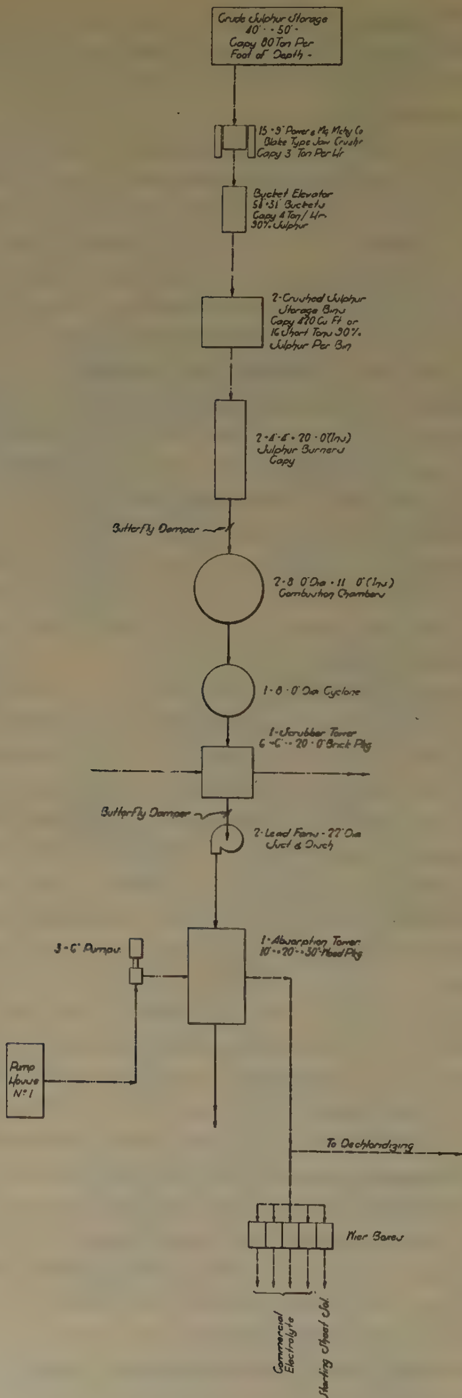


FIG. 18.—FLOW SHEET, SULFUR DIOXIDE PLANT.

supply the scrubber tower with 40 to 60 cu. m. The runoff from the two towers join and mix and are piped in gravity lines to a six-weir box unit in which the gassed solution is measured off for delivery to the electrolytic tank house and other vital points.

Over-all plant recovery is approximately 99 per cent. The plant capacity is 25 tons of fine sulfur per day, producing close to 50 tons of SO_2 gas of 14 per cent grade. The gassed electrolyte carries 0.7 to 2.0 grams per liter of SO_2.

Sulfur dioxide is not added to reduce ferric iron, and very little such reduction takes place in the operation as followed today, owing to time

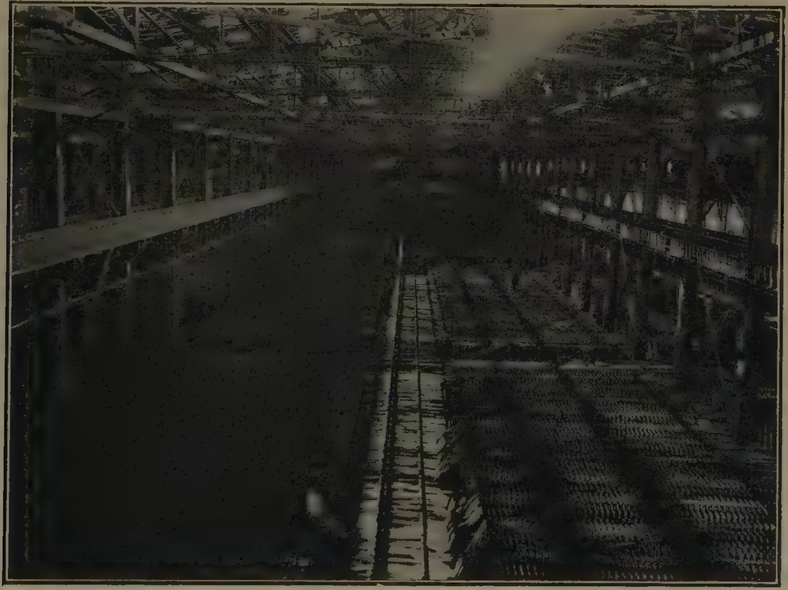


FIG. 19.—CENTER BAY OF ELECTROLYTIC TANK HOUSE. NOTICE GUAYAQUIL CANE ROOFING.

limitations. Approximately 90 per cent of the absorbed SO_2 is converted by anodic oxygen to sulfuric acid, and there is practically no odor of SO_2 in the electrolytic tank house with electrolyte containing 0.2 to 0.5 grams per liter of SO_2. The escape of SO_2 in electrolysis is prevented by the oil blanket used to cover the electrolyte in the cells.

Electrolytic Tank House

The tank-house building measures 1187 ft. long by 160 ft. wide, and is divided into three bays. The center bay is shown in Fig. 19. The floors and foundations are of reinforced concrete and the building itself is of structural steel, covered on the windward sides with corrugated iron or corrugated asbestos board, and roofed with Guayaquil cane and corru-

gated iron. This type of construction is possible because of the warm dry climate, and as in many other similarly constructed buildings in the plant, results in excellent ventilation. The concrete floors and column

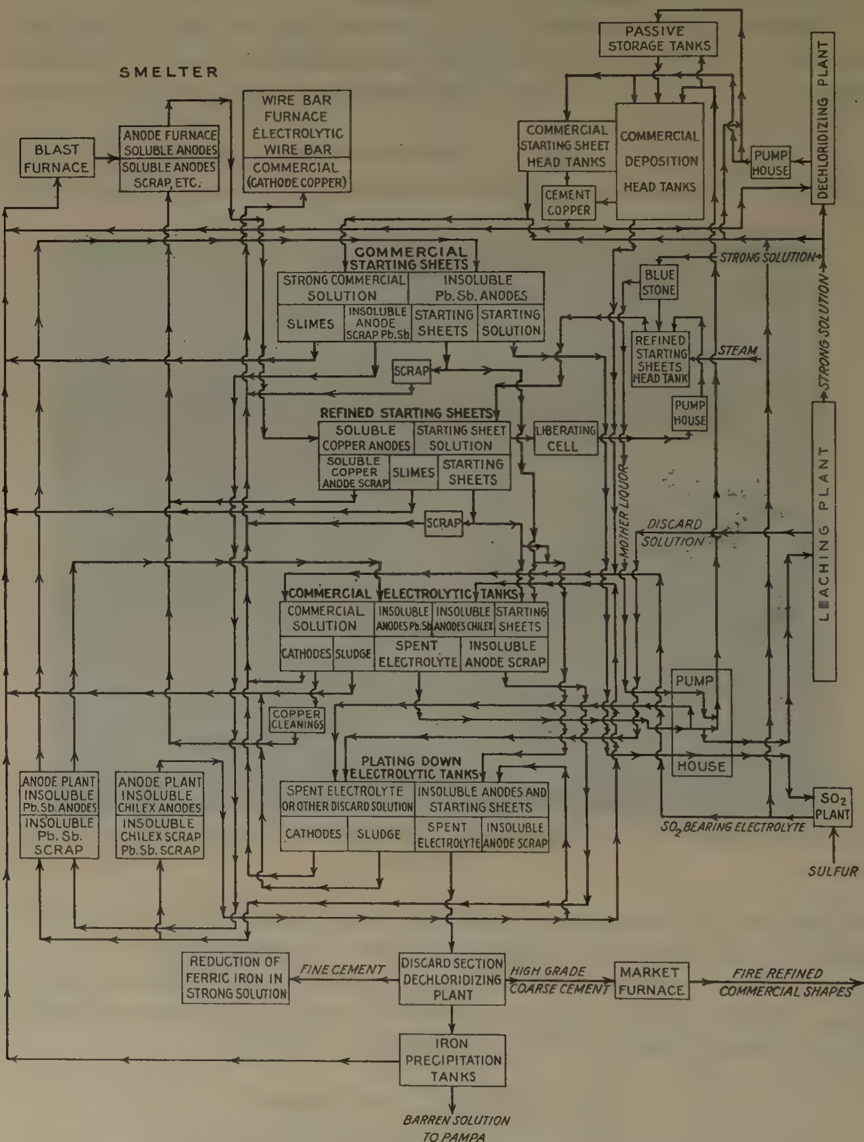


FIG. 20.—FLOW SHEET OF ELECTROLYTIC TANK HOUSE, 1933.

foundations are protected with mastic, and the structural steel by a mastic paint. Solution feed lines and runoff canals are in the basement, which is of ample height and the floor of which is pitched to drain spills and wash

water to a central sump. Power buses are carried on insulated rods, carried on the roof trusses. The substation, where power generated at the coast and transported to Chuquicamata on three high tension lines, is converted, is immediately adjacent and parallel to the tank house, with resulting short conductor leads and low lead loss.

There are 1098 electrolytic tanks in the building, each tank measuring 19 ft. 2 in. by 3 ft. 11 in. by 4 ft. 10 in. deep, inside dimensions. These tanks are constructed of reinforced concrete and are lined with mastic. Sixteen or seventeen tanks arranged in cascade comprise a section, and four to eight sections in series electrically make a circuit, of which there are ten in the house (Fig. 21). Each circuit is powered from one or more rotary converters or motor-generator sets (Fig. 22). This converting equipment is tied together by a system of buses, so that most machines can be cut in on any of the nine commercial circuits. One circuit and a variable number of tanks in one other circuit are used for the deposition of starting sheets, while one or more sections are always in use in the service of plating down solution to be discarded. The remainder of the house is engaged in commercial cathode deposition.

The deposition tanks rest on piers with ground footings and are constructed independently of the building proper. Tanks of the newer type are cast in unit blocks of 17 with but one inch pitch from inflow to outflow end, the direction of flow being across the short dimension of the tank. Tanks of the older type are cast singly and arranged in cascade, being connected by ports at alternate ends of the tanks, the direction of solution flow being with the long dimension of the tanks. The newer type of tank is much superior, as circulation is better, solution flow can be controlled more easily and with less danger of submerging anode lugs, and in the case of power failure the solution runoff to be disposed of is very small. The tank tops are 2 to 3 ft. above the floor level.

Four 10-ton cranes in each bay, or 12 in total, handle cathodes, anodes and starting sheets. Industrial track is laid in three longitudinal streets and in seven lateral streets. Winches at the heads of the lateral streets advance the cathode cars during the loading of cathodes, which is done by hand.

Adjoining the tank house and connected to it by industrial railway is the anode plant, where copper silicide (Chilex) and lead-antimony grid anodes are cast. The former are melted in three electric furnaces and are cast into cast-iron molds. Lead-antimony anodes are melted in oil-fired pot furnaces and the low melting point alloys are cast through valved outlets from the pots into cast-iron molds set on a revolving casting table. The capacity of this plant is 250 grid anodes per 24-hr. day.

The tank house is equipped with 60,000 anodes, part of which are Chilex and part lead-antimony. Starting sheets are made from lead-antimony anodes and dechloridized strong solution, and may also be made

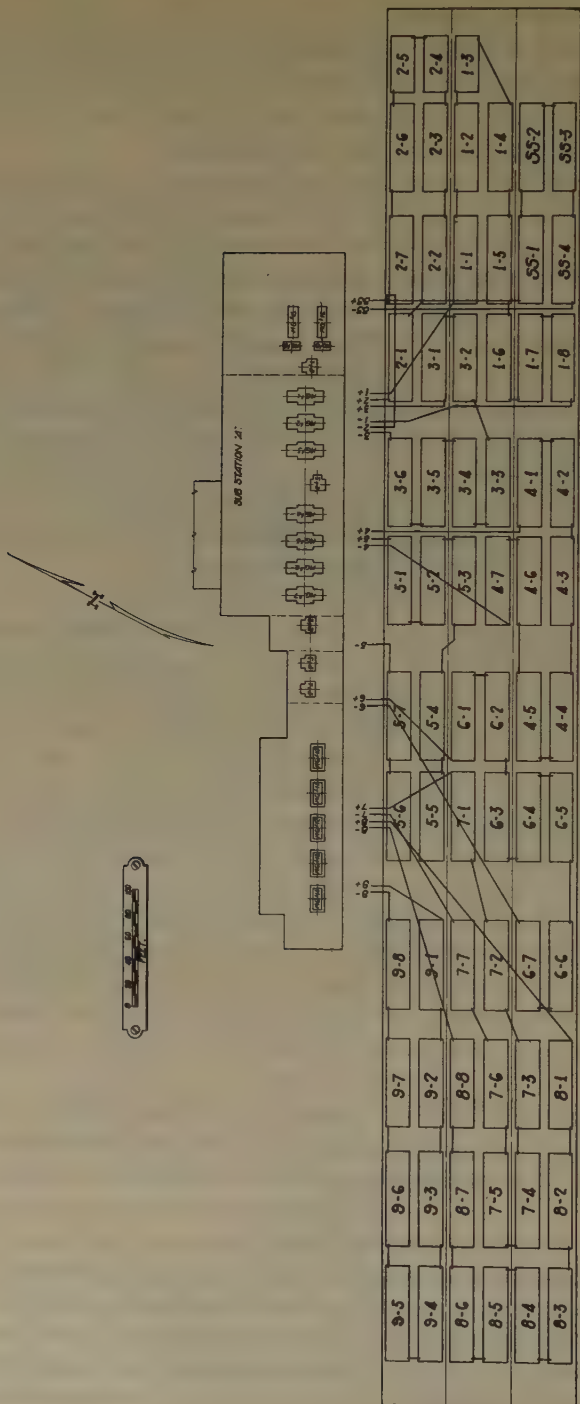


FIG. 21.—CIRCUITS, ELECTROLYTIC TANK HOUSE.

from soluble anodes and electrolyte made from bluestone, for which separate head tanks, dissolving tanks and pumps are provided.

Chilex anodes measure 2 ft. 9 in. by 5 ft. 11 in. by 1 in. thick, and are spaced on 3½ or 4-in. centers, depending upon operating conditions. Lead-antimony anodes of the same width and length but 0.5 to 0.6 in. in thickness are spaced on 3 or 3½-in. centers. Each cell therefore carries 56 to 74 anodes and 55 to 73 cathodes, depending upon the spacing. The lead-antimony anode produces approximately 12 per cent more deposition per kilowatt day than the Chilex anode and is gradually replacing the latter. The brittleness of the Chilex anode is a decidedly adverse factor

FIG. 22.—CONVERTING SECTION, SUBSTATION A, SUPPLYING POWER FOR ELECTRODEPOSITION FROM TRANSFORMED HIGH TENSION CARRIED FROM GENERATING STATION AT COAST.

in commercial operation with electrodes the size of those employed at Chuquicamata and the low iron and nitric acid content of present-day electrolyte is a favorable factor in the substitution of lead-antimony anodes. The cathodes from both anodes are of equal high purity. Warping of cast antimony-lead anodes under current is unknown here.

Commercial Starting Sheets

Starting sheets are deposited to 12-lb. weight in 24 to 48 hr. on copper blanks from freshly dechloridized and reduced strong solution, using lead-antimony anodes. Each tank in this service carries 72 anodes and 71 blanks and the cathode current density ranges from 7 to 15 amp. per square foot. The edges of the blanks are coated with a nonconducting grease and the entire sheet is stripped from the blank, the grooved edge not being used. Both wooden and steel tools are used in stripping the sheets from the blanks. All sheets are trimmed on all four edges prior to looping, in order to facilitate straightening before and during

deposition. The flow through each tank is independent and amounts to approximately 100 liters per minute.

The sheet as trimmed measures 3 ft. by 4 ft. 1 in., of which 3 by 4 ft. are immersed in electrolyte. The sheets are equipped with two loops cut from starting sheets, which are fastened to the sheet on Long and Alstatter machines. Approximately 8 per cent of the total sheets made are used for loops and about 2 per cent is trimming scrap. Blanks are amalgamated at intervals and are, of course, oil-coated before each deposition. The blanks used are made from oversize starting sheets built up to thickness by electrodeposition. They are trimmed and ground and riveted to the suspension bars, after which they are amalgamated and ready for use.

Chlorine in electrolyte over 0.2 grams per liter ruins commercial starting sheets, embrittling them so that they can be broken readily in the hand into small pieces, and looping such sheets is impossible. Ferric iron also will prevent deposition of good sheets. All starting-sheet anodes are equipped with spacers of insulating material to prevent short circuits. Anodes do not warp, but the copper blanks have to be straightened occasionally, and rolled lead cathode blanks which are sometimes used warp seriously.

Commercial Deposition

The outflow solution from the starting-sheet tanks is pumped to the SO_2 plant and is returned then to the commercial deposition section, a small portion joining the strong solution before and after dechloridizing, for the purpose of stabilization. The current density for commercial deposition ranges from 7 to 18 amp. per square foot of cathode surface, depending upon conditions. The inflow solution to the tank house is a mixture of this solution in the proportion approximately of 1 to 2 with the effluent of Chilex anode cells, which is high in ferrous iron and is not yet cut to spent grade. The mixture runs to the cells by gravity from the head tank through two main headers of 24-in. pipe.

Seven 12-in. diameter lateral loop lines in the basement with 4-in. risers and 4-in. valves bring the solution to each individual section. The outflow from the Chilex cells is a first-stage spent electrolyte and is returned to the head tank for mixing as described above. The outflow from the lead-antimony cells is highly oxidized, is of spent grade for leaching and is returned to the leaching plant. A portion of the latter solution, approximately 12 per cent, is bypassed to plating-down sections where it is further cut to suitable grade prior to complete stripping by the cuprous chloride reaction.

The electrolyte is therefore cut in three stages from strong solution to spent electrolyte, and in four stages to a lower than spent grade for stripping prior to discarding. Each volume of strong solution passes

through the tank house four times for the two stages of commercial deposition, once for the cut in the starting-sheet cells and once for the plating-down cut.

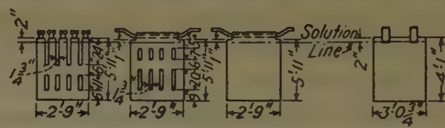
The flow is very fast in order to insure formation of hard cathodes and to decrease polarization. Each section takes a flow of approximately 750 liters per minute, although the maximum flow may be as high as 1200 liters per minute. The flow varies inversely with the grade of electrolyte and the current density. We do not exceed a cut of 8 grams per liter for one pass through 16 cells, or in other words 0.5 grams per liter per cell, whereas the minimum is as low as 2.0 grams per liter per 16 cells. The current efficiency drops off down the cascade as the electrolyte warms up and the ferric iron increases. The inflow solution carries 0.2 to 0.5 grams per liter SO_2 and the outflow from Chilex cells carries 0.05, while that from the antimony-lead cells has none. The depolarizing effect of the sulfur dioxide used is very slight, being equal to not over 0.01 volt.

The spent electrolyte temperature varies with the ampere load and atmospheric conditions from 30° to 45° C. We consider 36° to 39° the most efficient temperature for Chilex anodes and 34° for lead-antimony anodes. The latter, of course, have greatly superior anode efficiency and as a result do not heat the electrolyte as much as the Chilex anodes.

The electrolyte temperature is generally controlled to some extent by regulating the grade and therefore the volume of the strong solution. A spray pond is available also for cooling but seldom is required because the electrolyte more frequently requires heating than otherwise. The effect of hot electrolyte upon leaching extraction is a more important factor than its effect upon tank-house yield, when electrolyte is stable and the leaching cycle is short; i. e., under seven days.

With ferric iron in solution the inspection of cathodes becomes a very important part of the tank-house operation. Short circuits causing a dead cathode result in rapid resolution of the cathode. Starting sheets are straightened on inclined boards before loading into the sections, using a rubber-covered paddle. Within 8 hr. after loading and on the second and third days they are raised and again straightened. From then on the section is inspected twice daily for short circuits, which are spotted in the Chilex-anode equipped cells by a slight rise in the temperature of the cathode loops, and in the lead-antimony sections by raising the cathode about ½ in. by means of a hand hook, the short being detected by a tendency for the cathode to pancake to the anode. Shorted cathodes are raised and the sprouts cut off, the cathode reversed and returned to the tank. Deposition rapidly strengthens the joints where the loops are attached to the cathode, so that cathodes may be deposited to over 200 lb. if required. However, the average cathode weight employed here is 150 lb., which we find to be the most efficient weight for our conditions.

TABLE 9.—*Tank-house Electrodes*



Type anode	Chilex Grid	Pb-Sb Grid	Pb-Sb Solid	Cathode
Weight per anode, kg	140	150	160	Variable
Number per cell, 4 in. spacing	56	56	56	55
Number per cell, 3½ in. spacing	64	64	64	63
Weight per tank, 4 in. spacing, kg	7,840	8,400	8,960	
Weight per tank, 3½ in. spacing, kg	8,960	9,600	10,240	
Square feet under electrolysis, one side	8.71	9.27	10.54	12
Square feet under electrolysis, per tank, 4 in. spacing	958	1,020	1,159	1,320
Square feet under electrolysis, per tank, 3½ in. spacing	1,097	1,168	1,328	1,512
Current density, amp. per sq. ft. at 10,000 amp., 4 in. spacing	10.44	9.80	8.63	7.58
11,000	11.48	10.78	9.49	8.33
12,000	12.52	11.76	10.35	9.09
13,000	13.56	12.74	11.22	9.84
14,000	14.61	13.72	12.07	10.61
15,000	15.65	14.70	12.94	11.36
16,000	16.70	15.68	13.80	12.13
17,000	17.74	16.66	14.66	12.89
18,000	18.78	17.64	15.53	13.64
19,000	19.83	18.62	16.39	14.40
20,000	20.87	19.60	17.25	15.16
Current density, amp. per sq. ft. at 10,000 amp. 3½ in. spacing	9.11	8.56	7.53	6.61
11,000	10.02	9.42	8.28	7.27
12,000	10.93	10.63	9.04	7.93
13,000	11.85	11.13	9.79	8.59
14,000	12.76	11.98	10.54	9.25
15,000	13.67	12.84	11.29	9.92
16,000	14.58	13.70	12.04	10.58
17,000	15.49	14.55	12.80	11.24
18,000	16.40	15.41	13.55	11.90
19,000	17.32	16.26	14.30	12.56
20,000	18.23	17.12	15.06	13.22

Ampere load for current density of	at 4 in. spacing	at 3½ in. spacing
7	9,240	10,584
8	10,560	12,096
9	11,880	13,608
10	13,200	15,120
11	14,520	16,632
12	15,840	18,144
13	17,160	19,656
14	18,480	21,168
15	19,800	22,680
16	21,120	24,192
17	22,440	25,704
18	23,760	27,216

	Anode Thickness	Distance
Distance face to face electrodes at start of deposition, 3½ in. spacing	0.6	1.4
	1.0	1.2
4 in. spacing	0.6	1.65
	1.0	1.45

Short circuits, furthermore, raise the voltage of Chilex anodes upon resumption of the current, and cause the anode to deposit spotty cathodes. Raising the cathode for inspection is unfavorable to the Chilex anode for the same reasons. After a period of operation the voltage of the Chilex anode increases gradually and must then be restored by a process called "activation." This process, in brief, calls for alternate periods of 1 to 2 hr. with and without current flowing, the electrolyte being a weak sulfuric acid of approximately 10 grams per liter. This is carried on with 14 amp. per square foot current density for 12 to 24 hr., after which commercial electrolyte is run into the cells and normal operations resumed. While the voltage is materially reduced by this process and the deposition is more uniform and smooth afterwards, it is costly and therefore is not always justified.

The electrolytic tanks are covered with a petroleum oil known as smoothing oil, which is frothed mildly by the gassing of the electrolyte and forms a blanket over the solution, preventing largely the escape of spray and dissolved sulfur dioxide. There is considerable traffic on the tops of the tanks and this oil covering is a necessary feature to establish good working conditions and to protect equipment. The oil is held in the tanks by baffles. Because of the heavy traffic on the tanks a small attachment is used for the heads of the anode bars, which prevents the cathode bar from sliding off the anode bar when walked over. This prevents the edges of the Whitehead contacts from being rounded by arcing and results in a lower voltage drop over the contact, as well as preventing current losses and inconvenience due to shorts between bars.

The cathodes are deposited to 150 lb. in 5 to 15 days, depending upon the current density. At this weight they are pulled, loaded on to steel cathode cars equipped with acid and waterproof bearings and sent to the washing plant. They are then weighed and sampled and are ready for the market furnaces.

Cathode analysis is as follows: copper, 99.900; chlorine, 0.005; arsenic, 0.0002; antimony, 0.0004; lead, 0.0016; tin, 0.0011; selenium, none; tellurium, none; bismuth, none; iron, 0.0050; sulfur, 0.03; manganese, 0.0001; zinc, 0.0003; nickel, none; moisture, 0.02; insoluble (silica), 0.06.

Various data on the electrolytic tank house are given in Tables 9 and 10.

The Smelter

The smelter furnace equipment consists of three market furnaces for electrolytic copper shapes of 400 tons daily capacity each; one market furnace of 150 tons capacity for the direct refining of cement copper for the production of fire-refined shapes or for the casting of soluble anodes from furnace-refined black copper; and one blast furnace for the recovery

of copper from slag and miscellaneous secondaries of 50 tons fine copper daily capacity.

Table 10.—*Electrolytic Tank House Data*

Number electrolytic tanks in house	1,098
Number insoluble grid anodes to equip tanks	62,000
Current density, amp. per sq. ft. cathode area	7–18
Entering solution, grams per liter copper	21–26
grams per liter ferrous iron	1.6–2.1
grams per liter total iron	2.5
deg. C	26–34
Leaving solution, grams per liter copper	14–16
grams per liter ferrous iron	0.5
grams per liter total iron	2.5
deg. C	31–43
Current efficiency	85–92
Pounds copper per kilowatt-day	24–28
Capacity, kw. load	55,000

The 400-ton furnaces have a hearth length of 49 ft. 5 in. and a width of 13 ft. 10 in. The bottom and roof are of silica brick and the side walls are magnesite-lined. They are charged through three charging doors by Morgan open-hearth charging machines. Each furnace has a separate stack 6 by 6 by 75 ft. high constructed of silica and fire brick. The gases are cooled by a water jacket before entering the stack, but waste-heat boilers are not built in. The 150-ton furnace is equipped with magnesite sidewalls and bottom, while the roof is of silica brick. This furnace is equipped both for roof charging and side-door charging.

These furnaces are all oil-fired, the fuel being supplied at constant pressure of 75 lb. per square inch from two Traylor accumulators supplied from storage tanks through two Rumsey triplex pumps. The fuel oil is atomized with two Best burners on each furnace, utilizing 85-lb. air. Auxiliary air enters through a cast-iron nozzle below the burner and is supplied by a Sirocco blower of 15,000 cu. ft. per minute capacity at 10,000 ft. elevation. The auxiliary air is sucked under the furnace through built-in flues, which cool the brick lining of the furnace bottom. Excess air not required for combustion may be bypassed to waste.

A wire-bar furnace charge consists of hard, dense, pure cathodes plus rejected shapes, warmers, and skulls from the previous charge or charges. These metallics are expertly packed into the furnace by two charging cranes, after which the furnace is sealed and the melting operation started.

The fuel oil is burned by two burners and is gradually brought up to 460 gal. per hour. After 8 to 10 hr. the charge is flat and begins to boil, which is caused by the melting of cold metallics on the bottom of the furnace. After finish of boiling the charge is fully melted, at which time the fuel oil is decreased and the slag is skimmed clean. At this point

the copper contains 0.2 to 0.4 per cent of oxygen, most of which has been formed from the excess oxygen in furnace gases.

Iron pipes connected to 12-lb. air are then inserted through small openings in the side doors, poling door and bar recharging door. By means of these pipes air is blown into the molten metal to complete the oxidation of impurities. When approximately 0.8 per cent of oxygen is

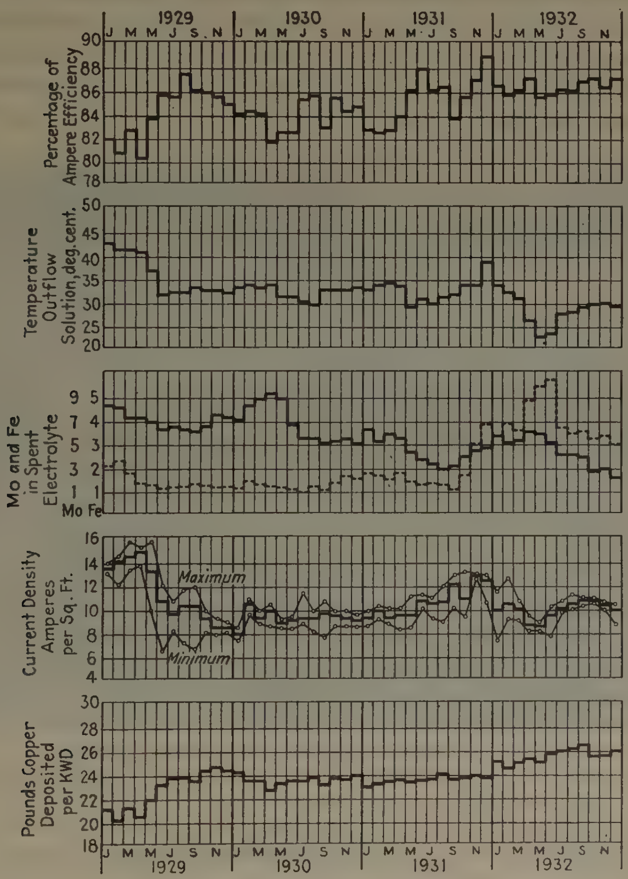


FIG. 23.—TANK-HOUSE DATA.

contained in the metal, which is evidenced by the crystalline character of test buttons, the charge is fully scorified and the slag is skimmed off. This slag as well as the previous one are oxidized silicate slags, the silica being furnished largely by erosion of the roof brick and the furnace bottom. No slag-making materials are added to the furnace ordinarily.

This slag is skimmed off clean, after which the charge is covered with a heavy blanket of low-sulfur coke and the reduction of copper oxide by means of poles forced into the metal bath is started. Very little fuel oil

is burned during this operation in order to keep the furnace atmosphere as free of sulfur as possible. The poling operation is followed closely by test samples and when the oxide has been reduced to 0.03 to 0.04 per cent and the pitch is right the furnace is tapped and casting starts. Each reverberatory furnace is equipped with a mechanical casting wheel, those of the 400-ton furnaces being of the Clark type. These wheels are 40 ft. in diameter and carry 30 molds of three to five pockets each supported on trunnions and arranged radially on the wheel. The wheels are of heavy cast-iron construction and are supported on inner and outer rollers. The drive is through a cable, which encircles the outer rim of the wheel and passes to a vertical cable drum driven through a worm drive from a 30 hp.

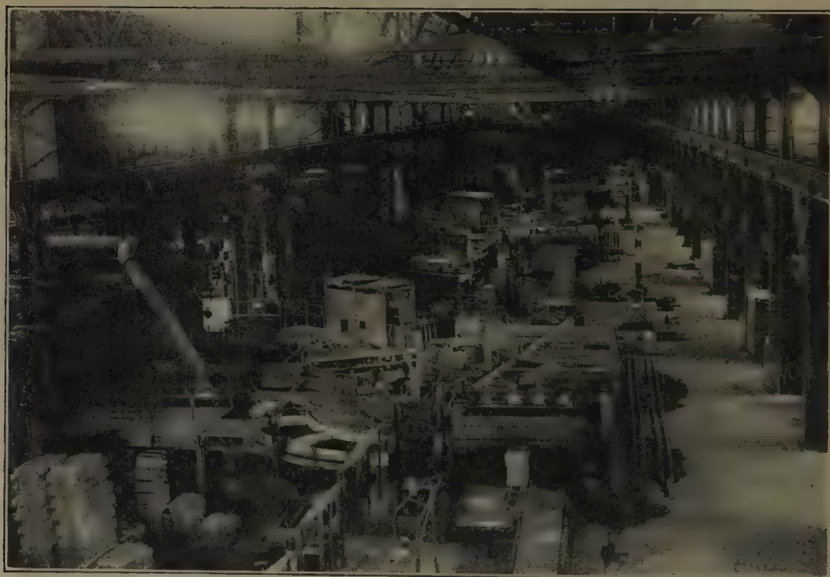


FIG. 24.—WIRE-BAR CASTING SECTION OF SMELTER.

Furnaces are in left bay and casting wheels are in pit. Shipping shed is at right and bosh and first inspection conveyors in center.

a.c. motor operated with automatic Cutler-Hammer control. Smooth starting, which is essential to the production of good shapes, is assured by the automatic starting equipment. The complete wheel weighs 164 tons. The 150-ton furnace has a Walker type casting wheel similar in principle but smaller and somewhat different in design. The Clark wheels will cast 50 to 80 tons per hour, depending upon the size and character of the shapes being cast. A general view of the casting section is shown in Fig. 24.

The furnace is tapped continuously into a divided launder that delivers the molten metal to two mechanical ladles, each of 2 tons capacity and formed with three to five equalized pouring lips. These ladles are hung

on yokes: the suspension point is well forward and the rear ends are elevated through a cable and pulleys actuated by hydraulic cylinders operating on 175 lb. water. The pouring movement is smooth and steady, simulating that of expert hand teeming.

The controls for the wheel and ladle operation are centralized in an enclosed bridge suspended over the casting wheel above and facing the ladles. Two operators are required on this bridge, each one handling one ladle and one operating the wheel also. In one ladle is suspended a specially designed fire end connected to a Brown recording potentiometer with magnified scale registering 1° F. change in metal temperature. Relays from this operate signal lights, which indicate two-degree changes in metal temperatures and are visible to the refiners from all parts of the operation.

Two molds are poured simultaneously, making a total of 6 to 10 bars poured at one time. Approximately 18 sec. is required for the actual pouring operation. While the bars are still molten the wheel is smoothly rotated 24°, centering the next pair of molds under the ladles. At a position approximately 90° from the pouring point, the bars have frozen and water is then applied by spray to the upper and under side of the molds. A metal hood connected to an exhaust stack is suspended over this side of the wheel and carries off the steam. At a point approximately opposite to the pouring point the molds are inverted and the bars fall into a bath of water, from which they are removed by a chain-driven variable-speed steel conveyor, which discharges them on to the first inspection conveyor on the main floor level. The molds while still inverted are further cooled by water spray applied to the bottoms. Approximately 72° from and approaching the pouring point they are righted and are then painted with a slurry of water and bone ash, using Murphy spray pistols. The mold at a temperature of approximately 95° C. dries this bone-ash coating thoroughly before the molds are again cast into. One round of the wheel requires about 10 min., depending upon the size of the shape being cast.

The molds are made from refined copper, the pockets being formed in the molten metal held in a heavy collapsible copper form by water-cooled copper cores fixed on the platen of an overhead hydraulic cylinder. When the metal has frozen, the cores are withdrawn, the form knocked down and the molds are removed. When cold they are cleaned and dressed, after which they are ready for use. Molds weigh approximately 2 tons each and are used for four to seven charges. Approximately 50 kilos of molds are required per ton of good shapes cast. Shrinkage and warping of molds is a problem, inasmuch as it involves variation in finished bar weight and dimensions and introduces hindrance to free shrinkage of the bar, particularly at the time of freezing, which may result in hair cracking of the hot shape. The application of cooling water

is directed in a manner designed to minimize mold warping. Mechanical casting of first class copper shapes demands faithful preparation of and attention to the numerous operating conditions and details. A change in the value of any condition invariably affects many other values, so that efficient casting can be realized only when all conditions are right simultaneously.

The cast bars are carried down the inspection conveyor, before which stand inspectors and helpers who discard defective bars and cut out small blemishes with a pneumatic chipping hammer. Each bar is gaged as it passes over this table. Rejects are remelted. The bars passing this table are inspected in great detail on a stationary table, each bar being personally viewed by the chief inspector. After leaving this second inspection the bars are made up into small drafts and weighed on the production bullion scales, after which they are stored in the shipping shed. Prior to shipping the bars are made up into lots varying from 5 to 100 tons and are reweighed on the bullion shipping scale. Shapes are handled by traveling crane and by electric tractor and trailers (Fig. 25).

FIG. 25.—TYPE OF ELECTRIC TRACTOR USED FOR HAULAGE OF COPPER SHAPES IN SHIPPING BUILDING.

Wire-bar analysis is as follows: copper, 99.9600; oxygen, 0.0320; sulfur, 0.0016; arsenic, 0.0002; antimony, 0.0004; lead, 0.0008; tin, 0.0006; selenium, none; tellurium, none; bismuth, none; iron, 0.0016; manganese, none; zinc, trace; molybdenum, 0.0001; nickel, none.

The refining of cement copper for the production of fire-refined shapes is carried out by a process similar to that employed in Michigan smelters on native copper concentrate. The operations are otherwise similar to the casting of electrolytic shapes.

Labor Efficiencies, Reduction Plant, Chile Exploration Co.

	Per Shift	Unit
Primary crushing	300	Short ton ore
Secondary crushing	112	Short ton ore
Leaching, dechloridizing, SO_2	190	Short ton ore
Tailings disposal	273	Short ton tailings
Tank house	1.85	Short ton cathodes
Insoluble-anode plant	6.0	Short ton cathodes
Smelting and melting	1.25	Short ton cast market shapes
Transportation	8.3	Short ton cast market shapes
Laboratory and sampling	5.8	Short ton cast market shapes
Total above operations	0.30	Short ton cast market shapes

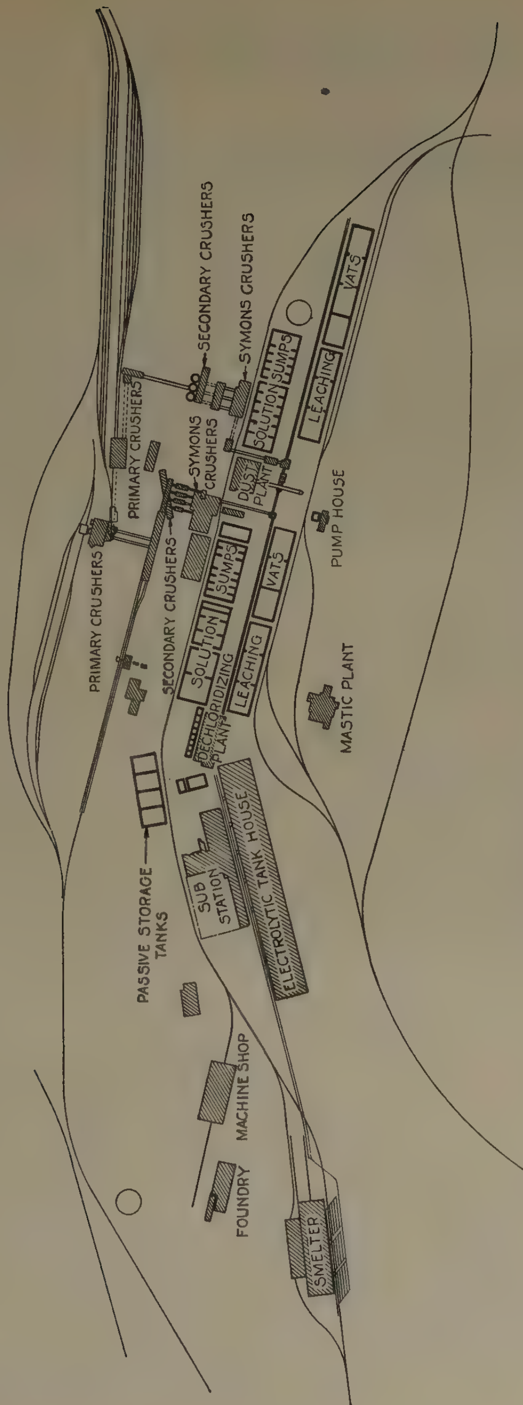


FIG. 26.—GENERAL PLAN OF THE PLANT.

TABLE 11.—*Smelter Data*

Short tons melted per furnace charge (wire bars)	360–400
Fuel-oil consumption per short ton (including heating up furnace), gal	23
Casting efficiency, per cent	85–90
Kilos molds per metric ton cast	50
Furnace cycle time, hours:	
Charging	1.5
Melting and skimming	14.0
Oxidation	3.0
Poling	1.5
Casting	5.0
Total	25.0

Development of the Leaching Operations of the Union Minière du Haut Katanga

By Archer E. Wheeler* and H. Y. Eagle,* New York, N. Y.

(New York Meeting, February, 1932)

The copper industry in the Province of Katanga, in the Belgian Congo, which is now controlled and operated by the Union Minère du Haut Katanga, had its inception many years ago in the vision of a Scotsman, Robert Williams, now Sir Robert Williams, Baronet. He it was who personally negotiated the mineral concession with King Leopold of Belgium, sent in the prospecting parties who located most of the present mines of the company, later arranged for the Belgian participation from which the present company was formed, and has been its active mentor ever since. The idea back of his efforts was the establishment of an industry on the Congo-Zambezi divide, of sufficient importance to draw up to it the head of the Cape to Cairo Railroad, the dream of Sir Cecil Rhodes, and the consummation of which he had left as a legacy to Mr. Williams, his successor.

The reports of the prospecting expeditions showed that copper in profitable quantity was there, but many difficulties lay in the way of the establishment of the infant industry. The isolated location of the mines, various tribal wars, the Boer War, and many other factors contributed to making progress very slow. However, in spite of these difficulties the railroad was finally built and a blast-furnace plant erected at Lubumbashi, near Elisabethville, the capital of the Province. Operations of this plant were started in 1911, treating high-grade oxidized ores and producing pig copper by a reducing smelting. Production has been continuous since that time and the company has grown so that at present it ranks as the largest individual copper producer in the industry. It is a fitting monument to the foresight, optimism and driving force of its founder, and the progressive policies of his Belgian colleagues.

In January, 1914, the Union Minière du Haut Katanga engaged as its consulting engineer, Archer E. Wheeler, metallurgical engineer, at that time general superintendent of the Great Falls plant of the Anaconda Copper Mining Co. Mr. Wheeler was commissioned by the Board of Directors to proceed into Katanga, and make all necessary investigations to enable him to report on a comprehensive development program, and to recommend a scheme of treatment particularly for their low grade

* Consulting Metallurgical Engineer.

ores. He reached Katanga in June, 1914, where he was joined by F. W. Snow, mining engineer, formerly connected with the Great Falls smelter and the Ray and Chino properties, whom he had engaged as his assistant.

The first step in the investigation was an extended inspection trip to the mines and prospects on the company concession. This involved a

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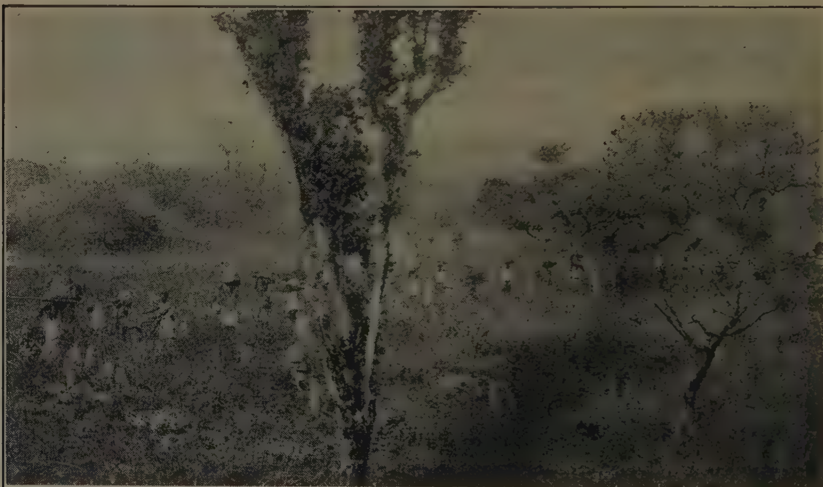

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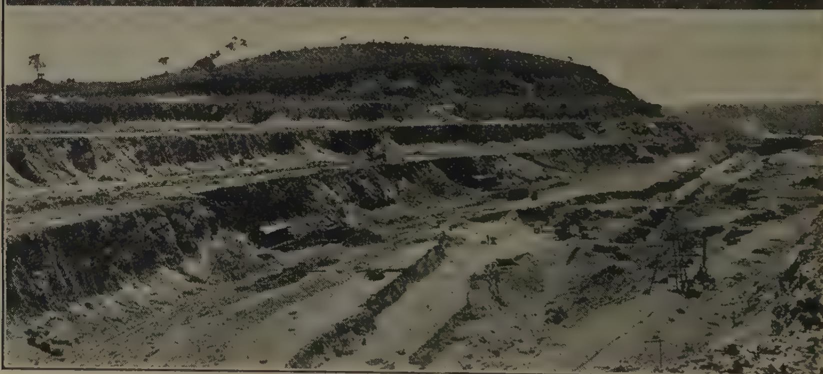

Fig. 1.—Inspection safari on trek.
Fig. 2.—Kambove, one of the mines of the Union Minière.

trek through the bush with a native safari of about 120 boys, and covered the period from Aug. 12 to Dec. 24. The trip involved a journey on foot (Fig. 1) and cycle of about 500 miles, and included visits to four operating mines and about 43 prospects. The mines were surveyed, mapped, described and sampled to the extent that outcrops and existing development and old native workings permitted. The mine at Kambove is shown in Fig. 2. The conclusions drawn from the results of the inspection trip can be summarized as follows:

1. It was estimated that the mines visited would yield an ore of an average grade of 6.5 to 7 per cent.

2. It was estimated that the amount of direct smelting ore which would result from mining operations would be small, probably not over 8 per cent if a grade of 15 per cent were required.

3. Practically all the ores were completely oxidized, the predominating mineral being malachite, with very minor amounts of azurite, chrysocolla, cuprite, native copper and occasional evidences of sulfides.

4. A remarkable similarity existed in the ores from the various deposits. The copper mineral was mainly found in shales, sandstones and quartzose association, which were siliceous in character.

5. It was obvious that any scheme of treatment recommended must be one to treat ores of a much lower grade than the existing Katanga practice of direct blast-furnace smelting of ores carrying 14 per cent or more of copper.

6. Sufficient ore was in sight in mines close to the railroad to justify the erection of a low-grade treatment plant.

7. The character of the ore occurrence and the lay of the ground at the various mines indicated that cheap steam-shovel mining could be applied in most cases for a low-grade treatment program.

The next step in the investigation was the study of a scheme of treatment, the methods naturally presenting themselves being:

1. Smelting of ores direct from the mine.
2. Water concentration, or flotation concentration, followed by smelting of the concentrates.
3. Leaching.
4. Combinations of any or all of these.

The desirable treatment, of course, was one to which all ores were commercially amenable, thus requiring only one type of plant and obviating all selective mining. As a necessary preliminary to a comparison of these methods, a thorough study was made of local conditions, and the development of unit costs to use in the estimates. This involved the following principal points.

1. Transportation costs on fuels, fluxes, ores, concentrates and copper.
2. Cost of white and native labor, including housing, traveling expenses and allowances.
3. Investigation of the available water power sites in the concession.
4. Cost of power from wood as fuel and coal as fuel.
5. Cost of power from the various possible hydroelectric developments.
6. Cost of operating and repair supplies.
7. Cost of mining, by various methods.

With these figures as a basis, complete direct smelting calculations and production costs were worked out for all the different types of ores from the various mines. From the results of these calculations, the general conclusion was drawn that ores to be profitable for direct smelting under the existing conditions should contain from 10.5 to 13.5 per cent

copper, depending upon the kind of ore. In order to furnish a reasonable margin of profit, somewhat higher grades than these would be required. In view of the probable small yield of this grade of ore from mining operations, it was obvious that direct smelting by itself did not furnish the solution of the problem.

Concentration and Leaching Tests

As no data existed as to the amenability of Katanga ores to concentration, it was necessary to do a great deal of experimental work along this line. The work was started in the laboratory on a very small scale, with hand-operated equipment. Later these were supplemented by tests on a larger scale run in a small test plant erected for the purpose, which provided facilities for hand picking, screening, jigging and tabling with the necessary intermediate crushing steps. Based on the test results, detailed smelting calculations and operating costs were worked out for the concentrates of various grades, as produced from the various ores. The general conclusions drawn from this work were as follows:

1. All the different types of Katanga ores can be enriched by water concentration, but generally with comparatively poor recoveries.
2. Most ores of a grade less than 12 per cent need to be enriched to be made to pay.
3. The best average grade of concentrate to be produced is around 25 per cent Cu.
4. The greatest economy in operation lies in concentrating the ore as close to the mine as possible.
5. Under the then existing conditions, the existing smelter at Lubumbashi was the most economical place to smelt the concentrates.
6. In the state of the art, as it then existed, oil flotation held out no promise for the concentration of Katanga ores.

In order to determine the possibilities of the leaching process, small-scale tests were first instituted in the laboratory. These included upward and downward percolation leaching tests, upward and downward percolation washing tests, bottle agitation tests and decantation washing tests. On the basis of the data obtained from these tests, a small experimental plant was built and the small-scale tests were supplemented by tests on a larger scale. The general conclusions drawn from the results of this work were as follows:

1. Downward percolation of the solution through the total ore is not possible with Katanga ores of any allowable size.
2. Downward percolation is possible if material finer than 200 mesh (0.074 mm.) is removed.
3. Upward percolation of solution through the total ore is not possible keeping the slimes in the upward percolation system.

4. Leaching by all agitation is indicated as the most feasible method for a simple flow sheet.

5. The copper in the company's oxidized ores is almost completely soluble in sulfuric acid solutions.

6. The strength of solution required to effect the solution is low, and the solutions can be used effectively at ordinary temperatures.

7. Any size of ore from 3 mesh (6.680 mm.)down can be treated in a reasonable time.

8. The time required for leaching is short; approximately 48 hr. for 3-mesh (6.680-mm.) material, and 9 hr. for 10-mesh (1.651-mm.) material being indicated.

9. The chemical consumption of sulfuric acid is not excessive, an average of 0.04 kg. per kilogram of gangue being indicated as an average for all ores treated.

10. Continuous counter-current decantation is indicated as the most feasible means of recovery of values in entrained solution in the tailings.

11. In general, it appears entirely practicable to treat all grades of the Katanga ores by sulfuric acid leaching, as far as getting the copper into solution is concerned.

In determining a method for recovering the copper from the leaching solutions, consideration was first given to various possible chemical precipitants for copper, such as scrap iron, sponge iron, lime, hydrogen sulfide and sulfur dioxide. Under the existing local conditions and the type of solution proposed, none of these held out any promise of furnishing a commercial solution of the problem. Accordingly, attention was concentrated on the recovery of the copper from solution by electrolysis with insoluble anodes. As soon as solution became available from the leaching tests, experiments were started on electrolytic deposition. Tests were run using (1) Antisell basket anodes (carbon); (2) Antisell coke-faced lead anodes; (3) plain lead anodes.

A long series of tests was run to determine the effect of depolarization of the anodes by introducing SO_2 gas into the solution. Tests were also run to determine the effect of impurities in the solution on the electrolysis and the probable practical limits of iron in solution. The general conclusions drawn from the test work were as follows:

1. Copper can be recovered readily from the sulfuric acid leaching solutions by electrolysis with insoluble anodes, and in a high degree of purity.

2. Carbon anodes of either the basket or coke-faced type disintegrate unless they can be completely depolarized.

3. Depolarization with SO_2 is partly effective with carbon anodes, but not sufficiently so to prevent them from disintegrating.

4. Introducing SO_2 into the solution results in very little depolarization of plain lead anodes.

5. Plain lead anodes are quite suitable for use in the solutions, and while the required voltage is high, ampere efficiencies are high, and the pounds of copper per kilowatt-hour are reasonable.

6. None of the impurities which accumulated in solution have any marked effect on electrolysis, except iron.

7. The presence of ferric iron in the electrolyte in any considerable quantity results in low ampere efficiency. When small in amount, its effect is not marked and a concentration of 3 grams per liter in the electrolyte was fixed as a probable practical operating condition.

Using the results of this test work as a basis, complete treatment costs by leaching were worked out for the Katanga ores of various grades. These were supplemented by comparisons of the costs by the various treatment schemes, from which it was concluded: (1) that leaching offers the most profitable method of treatment for the large bulk of Katanga ores; (2) that ores of a much lower grade can be treated at a profit by this method than by any of the other methods; (3) that leaching is the only scheme that does not involve a great deal of expensive selective mining.

The complete details of the results of the inspection trip, test work and calculations were embodied in a voluminous report which was completed and submitted to the Board of Directors in August, 1916. As a final conclusion to this report, Mr. Wheeler recommended the adoption, in principle, of leaching as the basis of the future treatment program, and the immediate installation of a plant of large capacity.

4000-ton Concentrator

While these investigations were in progress, the world war began, and it was necessary to move the company's head office from Brussels to London, where it was operated until after the armistice, in conjunction with the offices of the British Associates of the Union Minière du Haut Katanga, Messrs. Robert Williams & Co., now Sir Robert Williams & Co. The Board of Directors approved Mr. Wheeler's recommendations in principle and authorized him to get out the designs of a large treatment plant, together with estimated construction and operating costs. Since war conditions at this time rendered it impossible to procure in London the necessary engineering force for this work, Mr. Wheeler was commissioned to return to the United States and carry it out there. He and Mr. Snow established offices in New York in December, 1916, and assembled an engineering force under H. Y. Eagle, electrometallurgist, who had been associated with the Engineering and Electrolytic departments of the Anaconda Copper Mining Co. at Anaconda and Great Falls, Mont. Preliminary designs and estimates for a leaching-electrolytic plant to produce 30,000 metric tons of copper per year were immediately put in hand, and the estimated construction and operating costs submitted to the Board in October, 1917. War conditions, with

the attendant high prices of materials, freights and insurance, resulted in an estimated construction cost of some $20,700,000 for the treatment plant and necessary water-power development, and the board decided that this major project would have to be laid aside until the end of the war and until more normal price conditions prevailed.

At about this time it became apparent that unless steps were taken to provide additional sources of supply for the existing blast-furnace operations, the exhaustion of high-grade direct smelting ores would make

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FIG. 3.—THE 4000-TON PANDA CONCENTRATOR.
FIG. 4.—LOADING CONCENTRATES FROM BINS TO CARS.

it necessary to cut the company's output of copper. This, with the war going on and all the company's output being turned over to the British Ministry of War Munitions for war purposes, was not to be thought of. The matter of building a concentrator to treat ores below smelting grade and supply concentrates suitable for the smelter had been under consideration for a long time. The experimental work in Katanga had shown the project to be possible, though the recoveries were such that the method under normal conditions would represent abortive metallurgical

practice. In view of the necessities of the situation and the fact that tailings could be impounded for future leaching, Mr. Wheeler recommended, as a lower cost alternative to the leaching scheme, the installa-

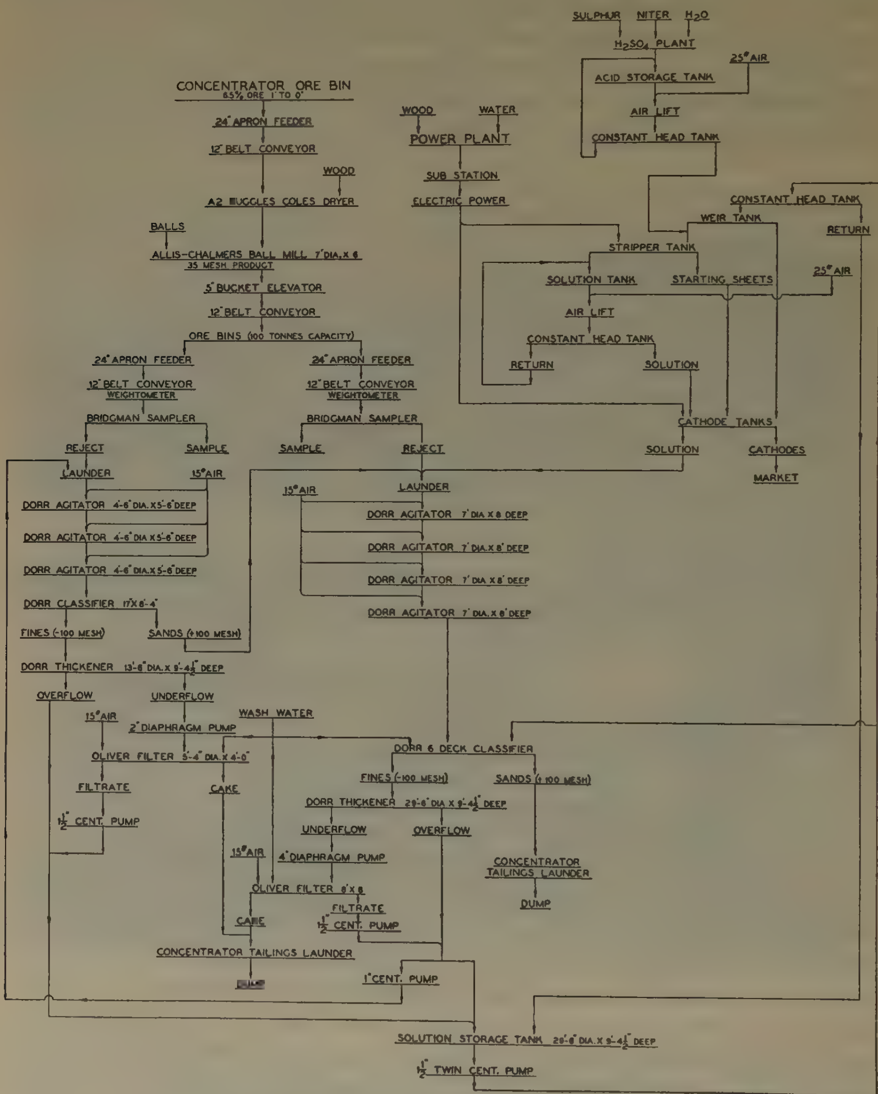

FIG. 5.—FLOW SHEET OF EXPERIMENTAL LEACHING PLANT.

tion of a 4000-ton concentrator at Panda, and the Board approved the project in November, 1917. The engineering force in New York was at once started on the work, and the purchase and shipment of material was accomplished as fast as war conditions would permit. The designs, purchase and shipment of material were completed by the end of

1918, construction carried out during 1919 and 1920, and the plant put into operation in the middle of 1921. (Fig. 3.)

Experimental Leaching Plant

Since the concentrator installation provided means for the desired expansion of the company's output for some years to come, time was afforded for a more leisurely consideration of the leaching project. The capital expenditure for the latter installation was so large that it was felt that a commercial demonstration of the process would be desirable to expedite its eventual financing. Mr. Wheeler, therefore, while the design of the concentrator was in progress, recommended the construction of the smallest unit that could use machines of commercial size of the type proposed for the large plant. The Board approved the design of such a unit in November, 1918, and authorized its construction in March, 1919. The work of design was started as soon as men became available from the concentrator design work, and the purchase and shipment of material was completed early in 1920. Construction was carried on in the field as fast as men became available from the concentrator construction force, and the plant was completed by the middle of November, 1921.

The crushing divisions of the concentrator were utilized to crush the ore for the experimental plant and the ore stored in, and drawn from, one end of the concentrator bins. The plant had a capacity of about 100 metric tons of cathode copper per month. It consisted of a drying unit, a dry-crushing unit, storage bins for ground ore, a wet-grinding unit, a leaching and filtration unit, an electrolytic tank house, and a small chamber sulfuric acid plant to furnish the necessary acid. Facilities for wet grinding in water were supplied as an alternate to dry grinding. Since this method involved an addition of water to the circuit, an evaporator was provided to remove this water from the leach solution. The fact that in practice the solutions normally ran at saturation in certain impurities rendered the evaporation scheme impractical, and this alternate was never used. The flow sheet of the plant as operated is given in Fig. 5. The principal features in which it departed from existing hydrometallurgical practice in copper are as follows:

1. The process is continuous, not batch.

2. Leaching is accomplished by agitation instead of percolation, this being dictated by the character of the Katanga ores.

3. Continuous washing of sand tailings is accomplished in multideck Dorr classifiers arranged counter-current.

4. Continuous washing of slime tailings on vacuum filters was installed on the basis of tests run by the manufacturer on Katanga ores. This was substituted for the original idea of installing counter-current decantation, one or the other of which methods was dictated by the character of the Katanga ore. The filtration method, if practicable, involved a much smaller initial capital expenditure than decantation.

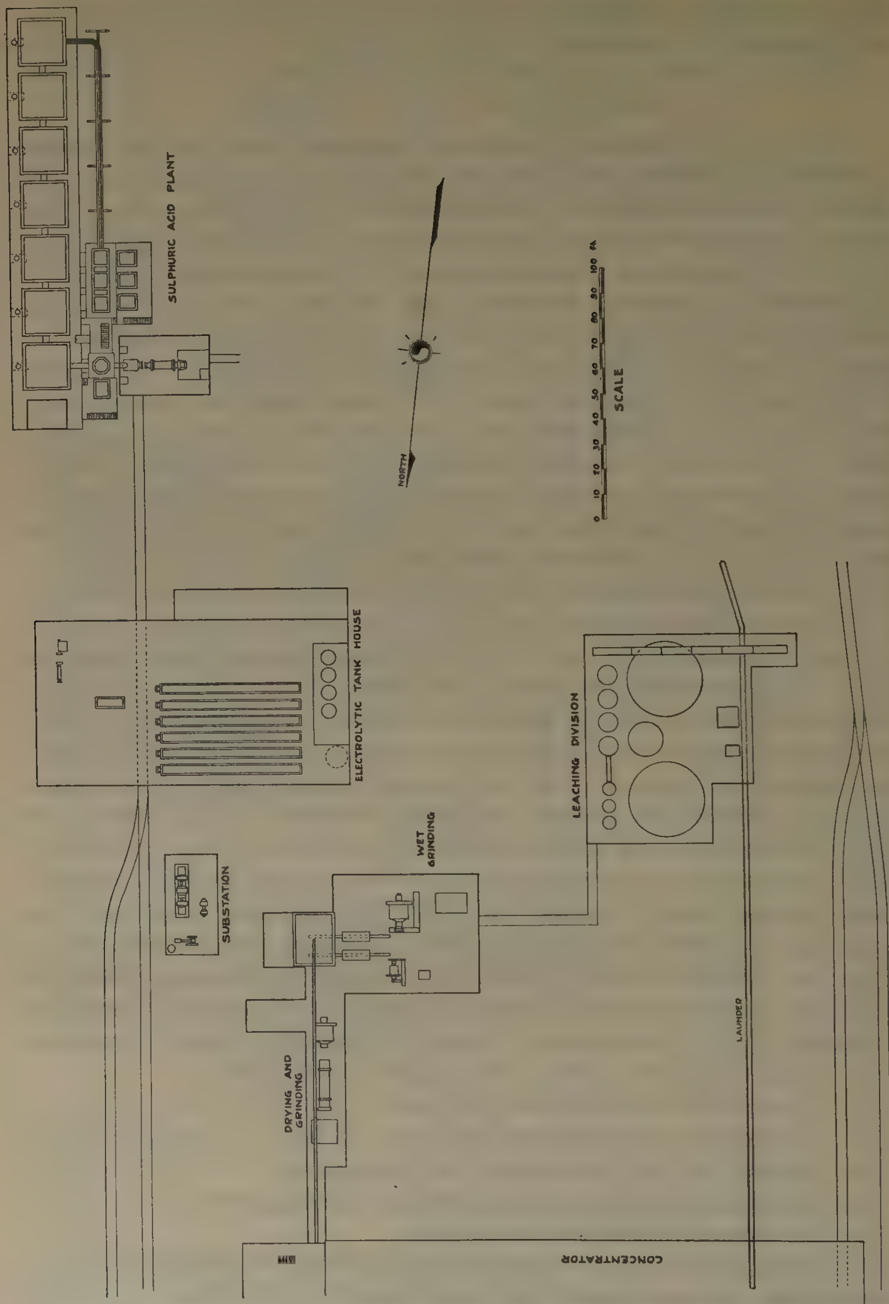

FIG. 6.—GENERAL PLAN OF EXPERIMENTAL LEACHING PLANT.

5. Provision was made for a continuous system of purification[1] by neutralization of a part of the solution with original ore or concentrates, to maintain the iron content of the solutions at a practical operating figure.

6. Electrolytic tanks[2] were installed which utilized in their design a principle not previously recognized.

The general arrangement of entire plant is shown in Fig. 6. A complete description of the electrolytic tank house is contained in another paper[3] and can be found elsewhere.

The experimental leaching plant was put into operation in November, 1921, and operated from then until the end of August, 1922, under the immediate direction of Mr. Eagle. Numerous difficulties were encountered in the early stages of the work, due primarily to the following causes: (1) inability of the agitators to handle the 35-mesh material promised by the manufacturer; (2) inability of the filters to handle the slimes tailings; (3) failure of much so-called acidproof equipment in the corrosive solutions.

The agitators would not progress the coarse material in the pulp and the accumulation of this material quickly plugged the first agitator in the series. After the system had been dug out a number of times, it became apparent that the agitators simply could not do the work. Experiments were then started with various expedients to progress the coarse material through the tanks. A simple attachment[4] for the agitators was finally developed, which did this so effectively that the fineness of the grinding was decreased to 20 mesh, and even small quantities of accidental coarse oversize up to 1/4 in. caused no trouble. Experiment proved the device effective for agitators of the Dorr or Pachuca types.

In spite of the fact that laboratory tests by the manufacturer with Katanga ores had indicated the vacuum filters as being capable of handling the slime, actual operation did not confirm the results of these tests. Excellent vacuum conditions could be maintained on the filters and the cake that formed did not crack, but after building up to about 1/8 in. in thickness, the filtration stopped. The washing of this cake, too, was ineffective, because of its impervious character. Since a cake of this thickness was not practical, it was necessary to revert to the original plan of counter-current decantation to effect the washing. The purification thickener was converted for a time into a washing thickener and run long enough to demonstrate the feasibility and effectiveness of this type of washing. During the test work, such slime tails as were not handled by the filters or washing thickener were discarded directly from the thickener spigots.

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[1] U. S. Patent 1703435.

[2] U. S. Patent 1431574.

[3] *Trans.* Amer. Electrochem. Soc. (1924) 45, 365.

[4] U. S. Patent 1651680.

The leaching solutions proved to be extremely corrosive, and copper, brass and bronze in contact with them went into solution with unbelievable rapidity. The bronze bolts in the leaching mechanisms, scrapers and valves of the filters, bronze filtrate pumps, etc., etc., all failed in quick succession, and had to be replaced by lead parts or steel parts covered with lead. The work developed the fact that only lead, duriron, rubber, asphalt mastic, glass and porcelain could safely be used in contact with the solutions. Later experiments added to the above list several alloys of the chrome-nickel-iron series.

TABLE 1.—*Details of Experimental Leaching Plant Operations*

QUANTITIES AND AVERAGE GRADE OF ORES TREATED

	Dry, Metric Tons	Cu, Per Cent	Cu, Metric Tons
Impregnated shales	3533.309	9.07	320.360
Talcose ores	324.875	7.97	25.891
Quartzose ores	1516.621	8.67	131.565
Fines from dumps	515.539	13.73	70.784
Total and average	5890.344	9.31	548.600

AVERAGE COMPOSITIONS OF ORES, PERCENTAGE OF DRY SOLIDS

	Impregnated Shales	Talcose Ores	Quartzose Ores	Fines Dump	Average Ore
Cu	9.07	7.97	8.67	13.73	9.31
SiO_2	63.95	60.57	68.17	56.19	64.17
Al_2O_3	8.48	6.98	4.85	7.93	7.42
Fe	3.40	4.73	2.45	3.23	3.21
CaO	1.32	0.89	1.30	1.36	1.29
MgO	4.88	9.22	4.52	5.13	5.05

TYPICAL AVERAGE SCREEN ANALYSES AFTER DRY GRINDING, INDICATING CHARACTER OF FEED TO LEACHING DIVISION, CUMULATIVE PERCENTAGE

Mesh	Impregnated Shales	Talcose Ore	Quartzose Ore	Fines Dump
+28	10.07	1.96	5.91	5.90
+35	16.93	4.87	13.69	11.74
+48	23.24	10.54	21.44	18.14
+65	29.65	17.63	29.62	26.46
+100	35.08	26.42	36.97	31.46
+200	48.98	40.22	51.85	44.87
Total	100.00	100.00	100.00	100.00

The various troubles encountered were gradually eliminated and the plant put on as nearly a regular basis of operation as the various lines of the experimental investigation permitted. The principal features of the experimental work are given in Table 1.

The copper content of all these ores must be rather startling to men accustomed to think in terms of the ores of the United States. The original survey of the company's mines had indicated that the average grade of ore would be approximately 6.5 per cent Cu, and that grade was requested for the experimental work. The mining department's attempts to meet the request resulted as shown in Table 1. One noteworthy effort was made to furnish the low-grade ore asked for, by mining a so-called low-grade ore and diluting it with barren material. Unfortunately, the barren material happened to be largely calcite and dolomite and when this material struck the leaching solutions, its effect can well be imagined. After this experience, straight ores were requested of as low a grade as the existing mining development permitted, and calculations depended upon to interpolate the results to the eventual average ore conditions.

As will be noted from the flow sheet, the tailings discharged from the plant are a sand tailing, an acid slime tailing and a purification slime tailing. Due to the inability of the filters to handle the slimes, and to the experiments with counter-current washing, the acid slime tails were discharged in three different ways during various periods of the investiga-

TABLE 2.—*Average Analyses of Tailing Products*

	Sand Tails	Acid Filter Cake	Acid Thickener Spigot	Washing Thickener Spigot	Purification Filter Cake
AVERAGE COMPOSITION OF TAILINGS, PERCENTAGE OF DRY SOLIDS					
Cu	0.331	0.245	0.176	0.169	2.40
SiO_2	84.20	68.56	75.41	74.62	64.91
Al_2O_3	4.65	10.36	8.77	9.01	11.40
Fe	2.65	4.14	2.99	3.40	3.94
CaO	1.20	1.32	1.17	1.31	0.88
MgO	3.08	7.26	4.72	5.86	5.47
PERCENTAGE OF WET PULP					
Solution per cent	30.25	31.79	50.61	50.96	40.83
ENTRAINED SOLUTION, CONTENTS IN GRAMS PER LITER					
Cu	1.46	16.11	34.34	24.09	31.08
Free H_2SO_4	0.12	4.90	4.30	0.96	0.00

TABLE 2.—(*Continued*)

AVERAGE SCREEN ANALYSES, CUMULATIVE PERCENTAGE

Mesh	Sand Tailings	Acid Slime Tailings	Purification Slime Tailings	Purification Classifier Sands
+ 28	9.13	0.01		9.22
+ 35	22.16	0.08		23.53
+ 48	39.47	0.25	0.05	39.76
+ 65	63.09	0.80	0.16	59.75
+100	78.29	3.01	0.29	73.96
+200	93.29	13.80	4.02	91.88
Total....................	100.00	100.00	100.00	100.00

tion: *viz.*, as acid filter cake, acid thickener spigot and washing thickener spigot. The average analyses of all the tailing products are given in Table 2.

The test results of the experimental plant indicated that the copper content of all the ores was readily soluble in sulfuric acid solutions, the average extraction (copper into solution) for all the ores treated being 97.44 per cent. An agitation time of 4 hr. was found to be sufficient to extract the copper from −20-mesh material. Inasmuch as the greater part of the slime tailings was discarded directly from the acid thickener spigot with no washing, the losses in entrained solution were very high and he over-all recovery was only about 80.31 per cent. The experiments and calculations on washing, however, demonstrated that this washing loss could be cut down by counter-current decantation to any desired figure which would justify the required initial capital expenditure.

During the experimental period, a total of 426 metric tons of cathode copper was produced. Table 3 shows averages of operating conditions.

TABLE 3.—*Averages of Operating Conditions and Analyses of Solutions*

426 Metric Tons Cathode Copper Produced

Cathode current density, amps. per sq. m....................	98.24
Cathode current density, amps. per sq. ft....................	9.13
Ampere efficiency, per cent..................................	83.33
Copper, kg. per kw-hr..	0.508
Copper, lb. per kw-hr..	1.120

TYPICAL ANALYSES OF SOLUTIONS TO AND FROM ELECTROLYTIC TANK HOUSE

	Free H_2SO_4	Cu	Fe, Total	Fe, Ferrous	Fe, Ferric	Al_2O_3	MgO	CaO
Solution from leaching plant, grams per liter...........	15	35	5	2	3	10	7	1
Solution return to leaching plant, grams per liter.....	60	15	5	3	2	10	7	1

The solutions at times carried considerable slime in suspension, owing to the slow settling properties of some particular ore, and some of this was mechanically entrapped in the deposited cathodes. The copper itself was of high electrolytic grade, and on subsequent furnace refining yielded wire bar of high quality. The slime content of the cathodes rendered them soft and brittle and unsuitable for regular shipment to Europe. For this reason fire refining of the cathodes was indicated as a necessary part of an eventual treatment plant in Katanga.

Tests in the purification section indicated that with ordinary copper ore in excess, an agitation time of 17 hr. in the purification agitators would give an effective elimination of Fe and Al_2O_3 from the leaching solutions. It was also demonstrated that the acid combined with the precipitated material was regenerated and used for dissolving copper. With the equipment as installed the purification tailings were rather high in grade, but tests showed that with proper control of the ore excess, and a finer point of classification on the purification classifier, this grade could readily be maintained below 1 per cent. This, in view of the small quantity of purification tailing produced, was considered quite satisfactory. Typical assays of solutions to and from the purification agitators are given below:

	Solution to Purification, Grams per Liter	Solution from Purification, Grams per Liter
Cu	35	56
Free H_2SO_4	15	0
Fe, total	5	2
Al_2O_3	10	6

The purification method used is not a new idea insofar as neutralization for precipitation of iron and alumina is concerned, but the experimental plant is the first, within the writers' knowledge, that has successfully applied it. The fact that ore in excess will precipitate iron and alumina from copper leach solutions has been known for a long time, but the loss of copper in the excess precipitant, and the re-solution of the precipitate in some subsequent step in the process, has always caused the idea to be discarded. The first difficulty was removed by classifying the pulp from the neutral agitators and sending the sands, which contain the great majority of the excess copper, to the acid circuit, where it is recovered. The second difficulty was eliminated by sending the neutral classifier overflow, which contains practically all the precipitate, to a separate thickening-washing circuit which is always maintained at neutrality. These two simple expedients rendered the scheme operative and furnished an effective means for controlling the iron content of the solution.

The consumption of sulfuric acid in the experimental plant confirmed the results of the earlier test work, the average chemical consumption for all the ores treated averaging 0.0376 kg. per kilogram of gangue. Due to the fact that most of the acid slimes were discharged from the acid thickener spigot with no washing, the entrained losses during the test period were high. As a result of this the total acid consumption for the entire period was 0.791 kg. per kilogram of copper recovered. The tests and calculations on counter current washing indicated that the washing loss could be cut down by this means to any desired figure which would justify the initial capital expenditure.

By the end of August, 1922, all necessary points bearing on the design of a large plant had been determined, the plant was at that time turned over to the operating management, and the consulting engineer's staff returned to New York. It is interesting to note that while the small plant was operated by the African organization from 1922 to 1929, no change was developed in the flow sheet which was submitted after the first nine months of work.

30,000 Metric Tons Panda Leaching Plant

The results of the experimental leaching plant were so satisfactory that the Board of Directors immediately authorized the final designs of a plant to produce 30,000 metric tons of wire bar per year, and this work was at once put in hand in the New York office. Estimates of construction and operating costs of the new plant were submitted in April, 1923, but the construction of the plant was not approved until March, 1925. A combination of circumstances then united to delay the construction of the plant, which could have been completed easily in two years. In the first place, the work was turned over to the company's newly developed African construction department, and, at the same time, the construction of a large reverberatory plant was authorized, the two projects to be carried on simultaneously. It was soon found that more had been undertaken than could be accomplished, and work on the leaching plant was finally stopped altogether until the reverberatory plant was completed, therefore the leaching plant was not ready to begin work until April, 1929, and was not really completed until the early part of 1930.

The flow sheet about which the 30,000 metric ton Panda leaching plant is designed is given diagrammatically in Fig. 7, and indicates the character of the equipment and the quantities of flow. The principal differences between this flow sheet and that of the experimental plant, which were developed by the test work, are as follows:

1. Ore bedding was substituted for bins for the storage of crushed ore. This was done both because Katanga ores are extremely difficult to handle in bins in the wet season, and also because it was desirable

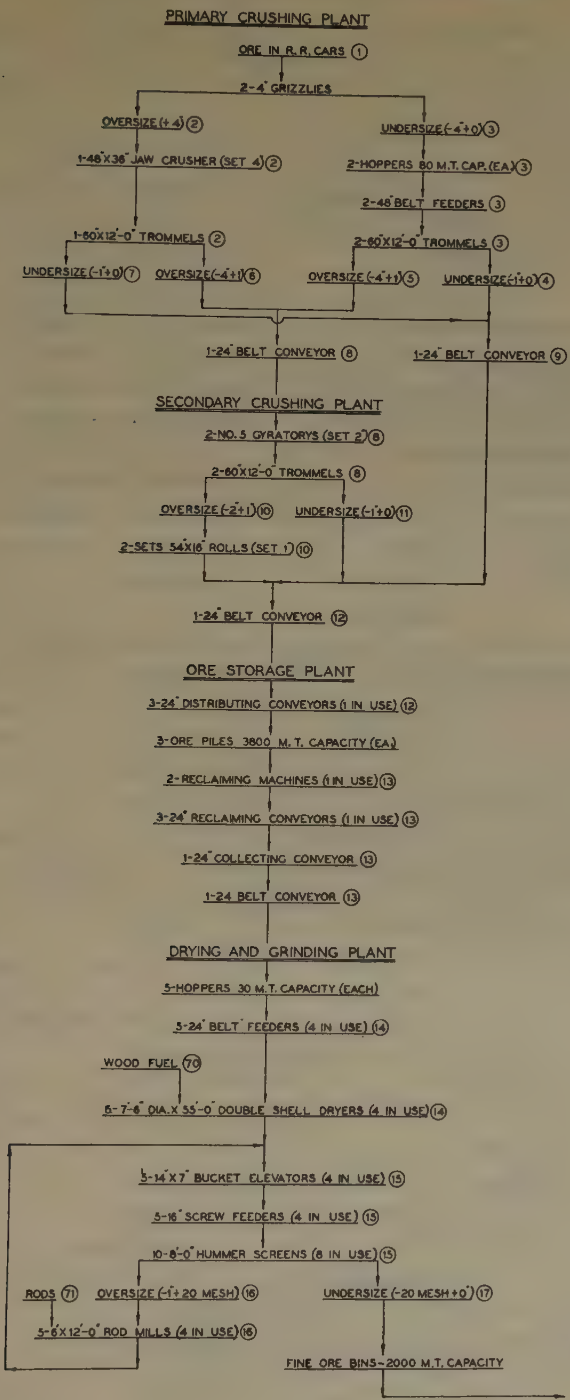
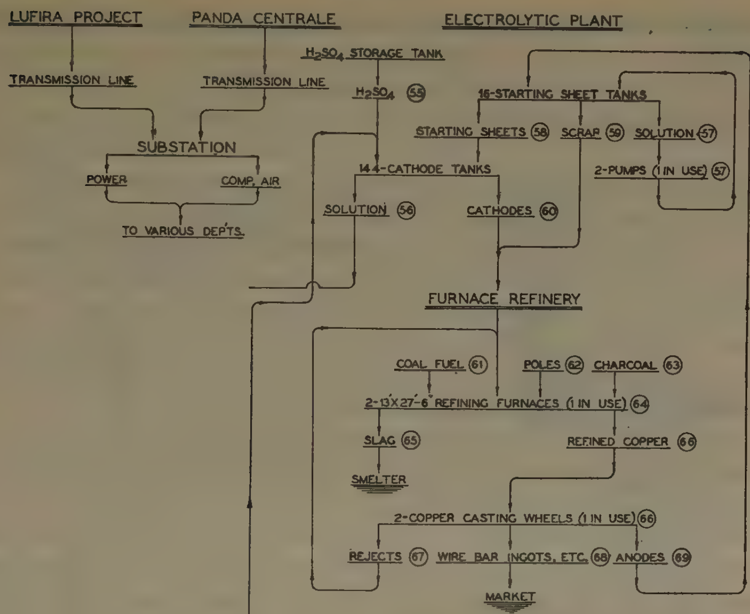

PRIMARY CRUSHING PLANT
ORE IN R.R. CARS (1)
2-4" GRIZZLIES
OVERSIZE (+4) (2)
UNDERSIZE (-4"+0) (3)
1-48"X36" JAW CRUSHER (SET 4) (2)
2-HOPPERS 80 M.T. CAP. (EA.) (3)
2-48" BELT FEEDERS (3)
1-60"X12'-0" TROMMELS (2)
2-60"X12'-0" TROMMELS (3)
UNDERSIZE (-1"+0) (7)
OVERSIZE (-4"+1) (6)
OVERSIZE (-4"+1) (5)
UNDERSIZE (-1"+0) (4)
1-24" BELT CONVEYOR (8)
1-24" BELT CONVEYOR (9)
SECONDARY CRUSHING PLANT
2-NO. 5 GYRATORYS (SET 2) (8)
2-60"X12'-0" TROMMELS (8)
OVERSIZE (-2"+1) (10)
UNDERSIZE (-1"+0) (11)
2-SETS 54"X16" ROLLS (SET 1) (10)
1-24" BELT CONVEYOR (12)
ORE STORAGE PLANT
3-24" DISTRIBUTING CONVEYORS (1 IN USE) (12)
3-ORE PILES 3800 M.T. CAPACITY (EA.)
2-RECLAIMING MACHINES (1 IN USE) (13)
3-24" RECLAIMING CONVEYORS (1 IN USE) (13)
1-24" COLLECTING CONVEYOR (13)
1-24 BELT CONVEYOR (13)
DRYING AND GRINDING PLANT
5-HOPPERS 30 M.T. CAPACITY (EACH)
5-24" BELT FEEDERS (4 IN USE) (14)
WOOD FUEL (70)
6-7'-6" DIA.X 55'-0" DOUBLE SHELL DRYERS (4 IN USE) (14)
5-14"X7" BUCKET ELEVATORS (4 IN USE) (15)
5-16" SCREW FEEDERS (4 IN USE) (15)
10-8'-0" HUMMER SCREENS (8 IN USE) (15)
RODS (71)
OVERSIZE (-1"+20 MESH) (16)
UNDERSIZE (-20 MESH+0") (17)
5-6'X12'-0" ROD MILLS (4 IN USE) (16)
FINE ORE BINS-2000 M.T. CAPACITY
TO LEACHING PLANT

Fig. 7.—(Part 1)

FIG. 7.—FLOW SHEET OF 30,000-



QUANTITY REQUIREMENTS ON THE BASIS OF TREATING A 6.5% ORE AND PRODUCING 30000 M.T. OF WIRE BARS PER YEAR

ORES

REF. NO.	MAX. DRY SOLIDS 24 PER HR. DAY M.T.	MAX. WET TONNES HANDLED BY MACH. PER DAY	MAX. WET TONNES HANDLED BY MACH. PER HOUR	HRS. OF OPERAT'N PER 24 HR. DAY	REF. NO.	MAX. DRY SOLIDS PER 24 HR. DAY M.T	MAX. WET TONNES HANDLED BY MACH. PER DAY	MAX. WET TONNES HANDLED BY MACH. PER HOUR	HRS. OF OPERATION PER 24 HR. DAY
1	1600	1860	235	8	11	600	632	85	8
2	400	421	75	8	12	1600	1860	235	8
3	1600	1860	235	8	13	1600	1860	235	24
4	1400	1628	220	8	14	1600	1860	100	24
5	800	842	120	8	15	3000	3060	170	24
6	200	211	30	8	16	1400	1429	70	24
7	300	316	50	8	17	1600	1633	70	24
8	1000	1053	150	8	18	300	306	15	24
9	1600	1860	235	8	19	1300	1327	60	24
10	400	421	65	8					

PULPS AND SOLUTIONS 24 HOUR SERVICE

REF. NO.	M.T. DRY SOLIDS PER HR.	LITERS SOLUTION PER HR.	PULP PER HOUR M.T.	PULP PER HOUR LITERS	REF. NO.	M.T. DRY SOLIDS PER HR.	LITERS SOLUTION PER HR.	PULP PER HOUR M.T.	PULP PER HOUR LITERS
20	57.6	243000	337.0	264700	39		267100		
21	50.4	250200	338.1	269200	40		267100		
22	33.0	243000	312.3	255400	41				
23	17.4	7300	25.8	13800	42		25400		
24		7300			43		241700		
25	17.4	14600	32.1	21100	44	12.5	25400	41.7	30100
26		7300			45	8.0	24100	35.7	27200
27	17.4	7300	24.7	13800	46	3.4	1300	4.9	2600
28	33.0	278000	352.7	290400	47	8.0	34500	47.7	37500
29		243000			48		24100		
30	33.0	35000	73.2	47500	49	8.0	10400	19.9	13400
31	33.0	70000	113.5	82500	50	8.0	20800	31.9	23800
32		35000			51		10400		
33	33.0	35000	73.2	47500	52	8.0	10400	19.9	13400
34		35000			53		155600		
35	58.4	208300	273.4	230300	54		10400		
36		243000			55		2000		
37		243000			56		241700		
38					57		36000		

MISCELLANEOUS

REF. NO.	METRIC TONNES PER YEAR	METRIC TONNES PER DAY	REF. NO.	METRIC TONNES PER YEAR	METRIC TONNES PER DAY	REF. NO.	METRIC TONNES PER YEAR	METRIC TONNES PER DAY
58	3150	9	63	350	1.0	68	30000	85.7
59	1050	3	64	39900	114.0	69	4200	12.0
60	33600	96	65	900	2.6	70	70000	200.0
61	8750	25	66	39450	112.7	71	175	0.5
62	1750	5	67	5250	15.0	55	28700	82.0

TON PANDA LEACHING PLANT (Parts 2 and 3).

that the grade of the ore fed to the leaching plant should be as uniform as possible.

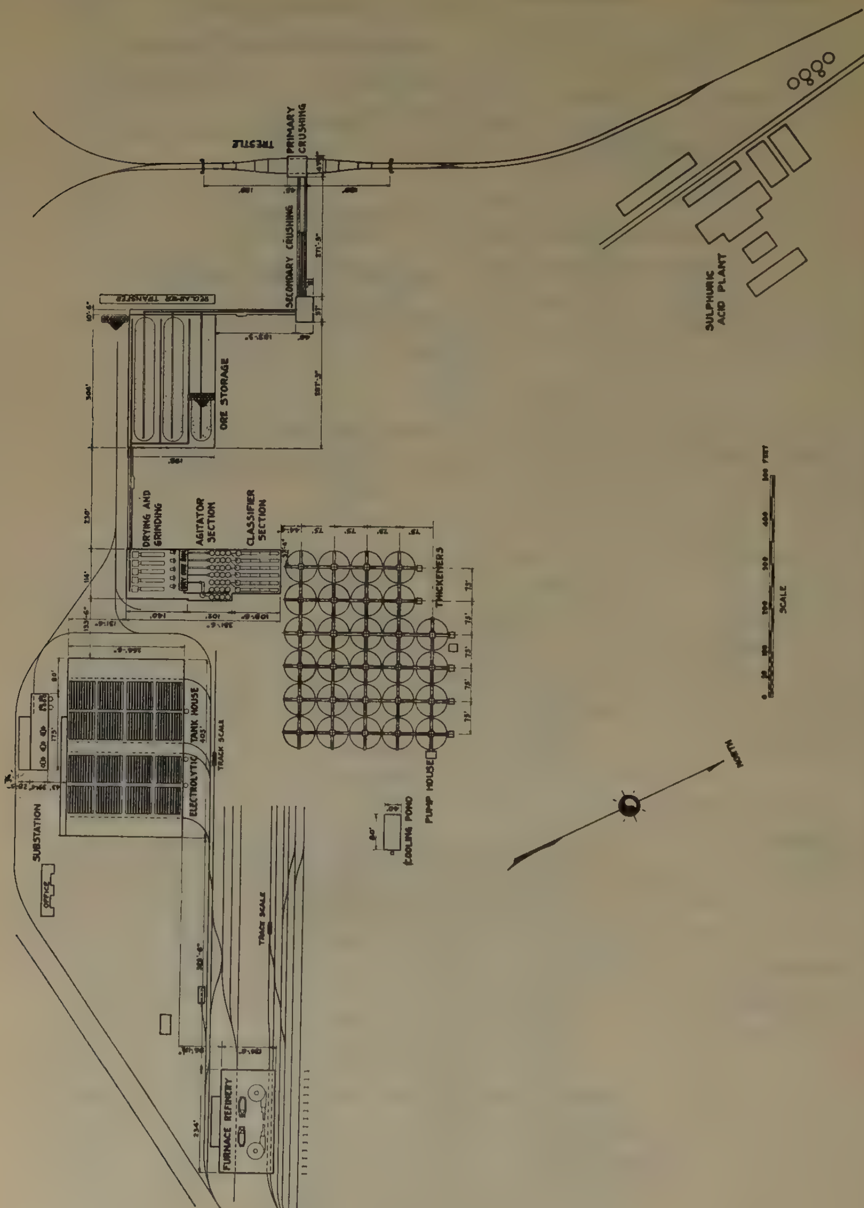


Fig. 8.—General plan of 30,000-metric ton Panda leaching plant.

2. Rod mills were substituted for ball mills for dry grinding and the fines scalped from the feed to the mill. Both these features were adopted to increase the amount of sand tailings from the ore. This was desirable

on account of the higher washing efficiency obtained with sands in the classifier than with slimes in the thickeners.

3. The primary classifiers in the acid section were made of the bowl type instead of standard, in order to increase the amount of sand tailings, for the reasons given above.

4. The purification classifier was made of the bowl type to increase the amount of sand discharge, and to attain the consequent lowering of grade of the purification slimes.

5. Counter-current washing was installed for recovery of solution values in entrained solution in slimes tailings. Three washes were decided upon for the initial installation, but the plant so laid out that more could be added in the future if conditions warranted it.

6. The electrolytic tanks were made somewhat longer but were arranged with three electrical groups instead of five.

7. Starting sheets were made from soluble anodes in a separate solution circuit, instead of with insoluble anodes from the leaching solution. This was done because it was a more certain method of regularly making satisfactory sheets, and also because it permitted the lowering of the grade of the solutions in the leaching division and thus decreased the washing losses.

8. A furnace refinery was installed to convert the cathodes into wire bar, to make the starting sheet anodes, and incidentally to furnish a product quite suitable for shipment to Europe.

A short description of each of the departments of the plant is given below. Fig. 8 shows a general plan. The plant is substantially built throughout, of good materials. The construction lumber is Pacific Coast fir; brick and cement are of the company's local manufacture; structural steel and most of the machinery not specifically identified are of Belgian manufacture. The plant is arranged so as to permit it to be extended on the present site to double its initial capacity of 30,000 metric tons of wire bar per year.

Crushing and Grinding

The primary crushing building is a steel-frame structure 45 ft. 0 in. by 47 ft. 2 in. in plan, sheathed and roofed with corrugated galvanized steel. It has four operating floors. The character of the crushing and screening equipment, which is of Allis-Chalmers manufacture, is indicated in the flow sheet (Fig. 7). The building is served by a double-track steel trestle 56 ft. 5½ in. maximum height by 378 ft. 0 in. long, by which the ore is brought to the receiving hoppers in standard (3 ft. 6 in. gage) drop-bottom railroad cars. A 20-ton hand-power crane over the crusher provides facilities for heavy repairs.

The secondary crushing building is a steel-frame structure, 40 ft. 0 in. by 57 ft. 0 in. in plan, sheathed and roofed with galvanized corrugated

steel and arranged with five operating floors. The crushing and screening equipment is as noted on the flow sheet and is of Allis-Chalmers manufacture. I-beam crawls and Yale & Towne trolleys and chain blocks over the various machines furnish means for heavy repairs. Conveyors bring the ore from the primary crushing and deliver the crushed product to the ore-bedding system.

The ore-storage building is a steel-frame structure 198 ft. 0 in. by 304 ft. 0 in. in plan, roofed and sheathed with galvanized corrugated steel and provided with two main operating levels. It consists of three bays, each housing an ore pile which is built up by the discharge of the self-propelling, self-reversing tripper on the conveyor above. Ore is reclaimed from the pile by a standard Messiter reclaimer, as manufactured by the Robins Conveying Belt Co., and is delivered by conveyor to the dryer bins. One pile is being built, one being fed, and the other standing by at any one time. A transfer car is provided to shift the reclaimers from one pile to the other. The conveyors in this division as well as throughout the rest of the plant, are equipped with ball-bearing idlers of Stephens-Adamson manufacture, and the belting was supplied by the Goodyear and Goodrich companies. Weightometers, from the Merrick Scale Co., are provided wherever necessary, to indicate and record the tonnage handled by the conveyors.

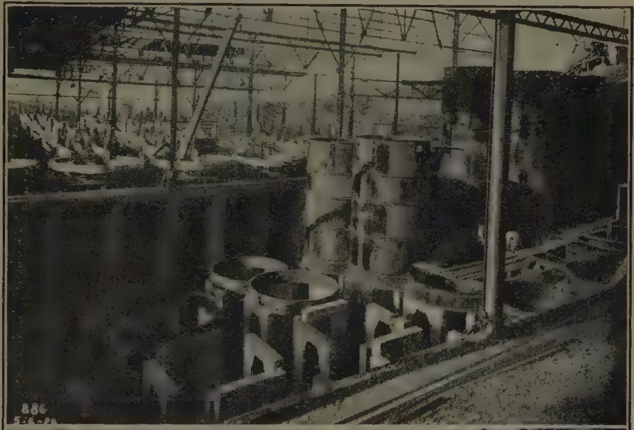
The drying and grinding building is a steel-frame structure, 114 ft. 0 in. by 140 ft. 0 in. in plan, sheathed with corrugated galvanized steel, roofed with wood decking covered with built-up roofing, and arranged with four main operating levels. The character of the equipment is indicated in the flow sheet, the dryers being furnished by the Ruggles-Coles Mfg. Co., the rod mills by the Allis-Chalmers Mfg. Co., and the vibrating screens by the W. S. Tyler Co. The rod mills are provided with individual synchronous motor drive through a Cutler-Hammer magnetic clutch. A 20-ton hand-power traveling crane over the mills provides facilities for heavy repairs.

The bins, holding 2000 metric tons, are of heavy steel construction, 20 ft. 6 in. by 95 ft. 0 in. in plan by 35 ft. 0 in. high. The bin bottom is of the catenary suspension type. An exhauster fan maintains the atmosphere in the bin at a small negative pressure, and this eliminates to a great extent the dust nuisance from the vibrating screen operations.

Leaching Division

The agitator building of the leaching division is a steel-frame structure, 102 ft. 0 in. by 115 ft. 6 in. in plan, sheathed with asbestos-protected metal, roofed with wood decking covered with built-up roofing, and provided with three main operating levels. Agitators are of the standard, cone-bottomed Pachuca type, built of steel and lined with 10-lb. chemical lead sheet. They are equipped with the device developed to progress

coarse material through the tank, and provided with an air supply of about 30-lb. gage, controlled from the main operating floor. The agitators are arranged in five acid sections of four in series, and one purification section consisting of two rows of four in series, arranged in parallel. The solution from the electrolytic tank house flows to the agitator build-



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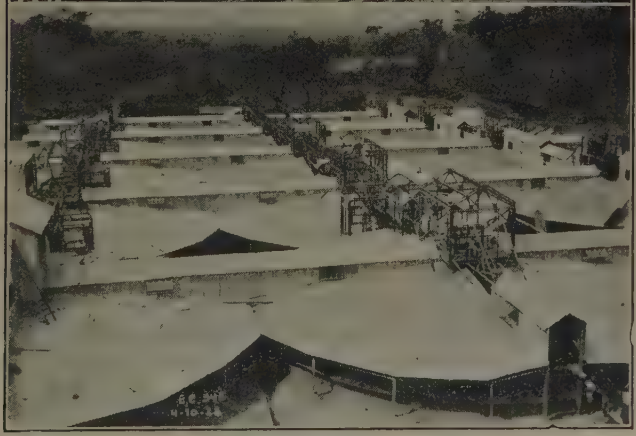

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FIG. 9.—AGITATORS AND CLASSIFIERS UNDER CONSTRUCTION.
FIG. 10.—THICKENER SECTION UNDER CONSTRUCTION.

ing by gravity and is distributed to the five acid sections by means of a five-compartment weir box. The solution to each section joins the ore feed to that section in a mixing box immediately ahead of the first agitator in the series. The solution feed to the purification section is pumped from the first storage thickener in an amount necessary to maintain the total iron content of the solutions at approximately 6 grams per liter. Fig. 9 shows agitators and classifiers under construction.

The classifier building of the leaching division is a steel-frame structure, 109 ft. 6 in. by 99 ft. 0 in. in plan, sheathed with asbestos-protected

metal, roofed with wood decking covered with built-up roofing, and provided with two main operating levels. The classifiers are arranged in six sections, five of which correspond to the five acid agitator sections, and one spare. Each section consists of a primary bowl classifier, followed by and built integral with a three-deck washing classifier. The sands are progressed successively through the four decks, which are arranged for counter-current washing, and the discharge from the last washing deck is the finished sand tailing. Water is added in the last washing deck in an amount equal to the entrained solution in the sands from this deck. The bowl-classifier overflow is laundered to the primary-thickener feed wells. The classifier tanks are constructed of reinforced concrete lined with 2 in. of asphalt mastic. The mechanisms are of steel, and where in contact with solution, are covered with 12-lb. chemical lead sheet. The rake blades are of duriron.

The thickener section of the leaching division consists of twenty-eight 70-ft. dia. Dorr thickeners, arranged on 75-ft. centers in both directions, in five horizontal benches, each bench 3 ft. 4 in. lower than the one above. There are six thickeners in each of the four top rows and four thickeners in the bottom row. The top row is the third wash, the next lower row is the second wash, the next lower row is the first wash, and the next lower row is the primary thickener. The first row of thickeners on the left looking up the hill is the purification section. The other five rows of thickeners are acid thickener sections corresponding to the same division in the agitator and classifier sections. One thickener in the bottom row is a spare used to hold the contents of any thickener that must be emptied for repairs; one is a clarifying thickener, which receives the overflows of all the primary acid thickeners, and the other two are simply solution-storage tanks. Each thickener is roofed individually and walkways are provided over the top connecting all the thickeners. This superstructure is of steel framing and the roofing of corrugated galvanized steel; walkway floors are of wood. The thickener tanks have concrete bottoms, steel sides and are lined with 8-lb. chemical lead sheet. Under each row of thickeners up and down the hill is a concrete tunnel which provides access to the cast-lead discharge cones, valves and piping, and which terminates in a sump and pump house. The latter provides facilities for disposal of pulp and solution drained from the tanks. The thickener mechanisms are constructed of steel with wood extension arms, the steel being covered with 12-lb. chemical lead sheet wherever in contact with the solution. Duplex Dorrco pumps progress the spigot discharge up the hill from one tank to the next, and the solution overflow flows by gravity from one tank to the next. Wash water is added in the last washing thickeners in an amount equal to that discarded as entrained solution in the finished slime tailing from this thickener. The feed to the five acid thickeners is distributed by a five-

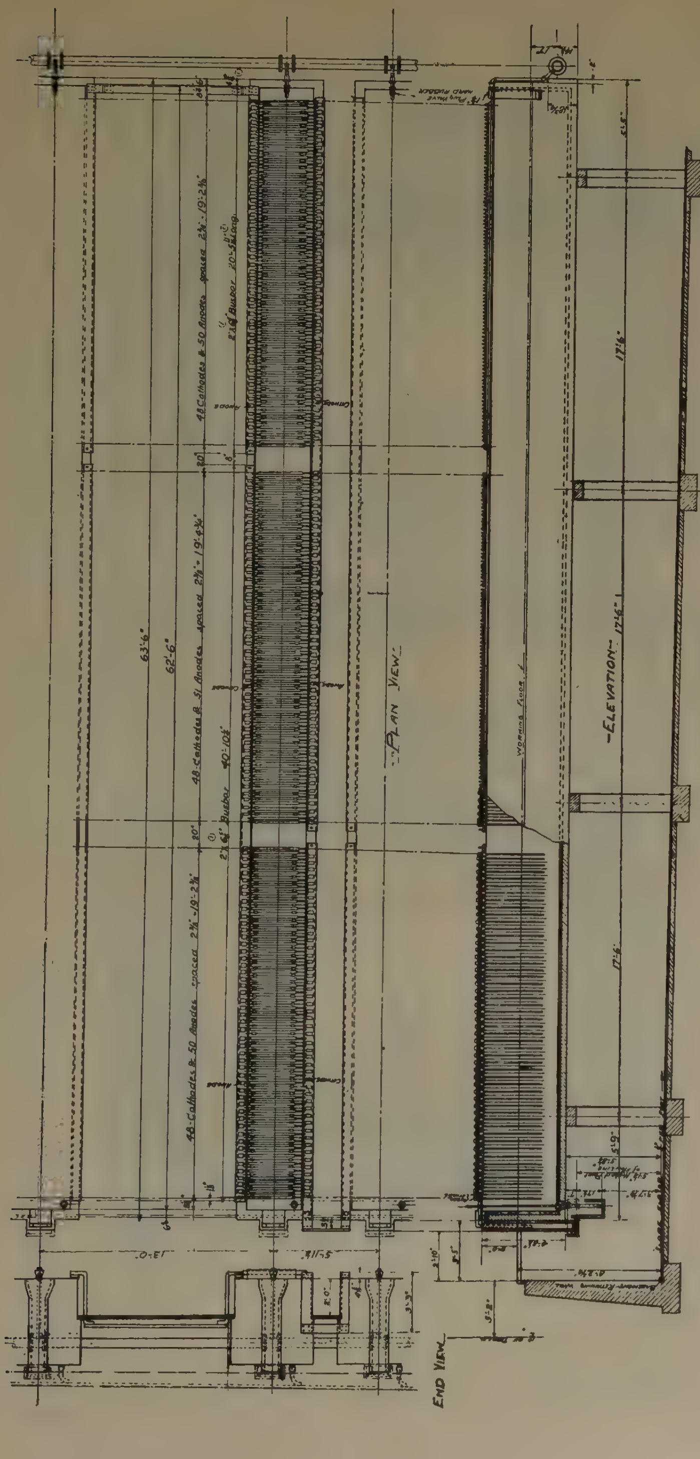

FIG. 11.—(PART 1)

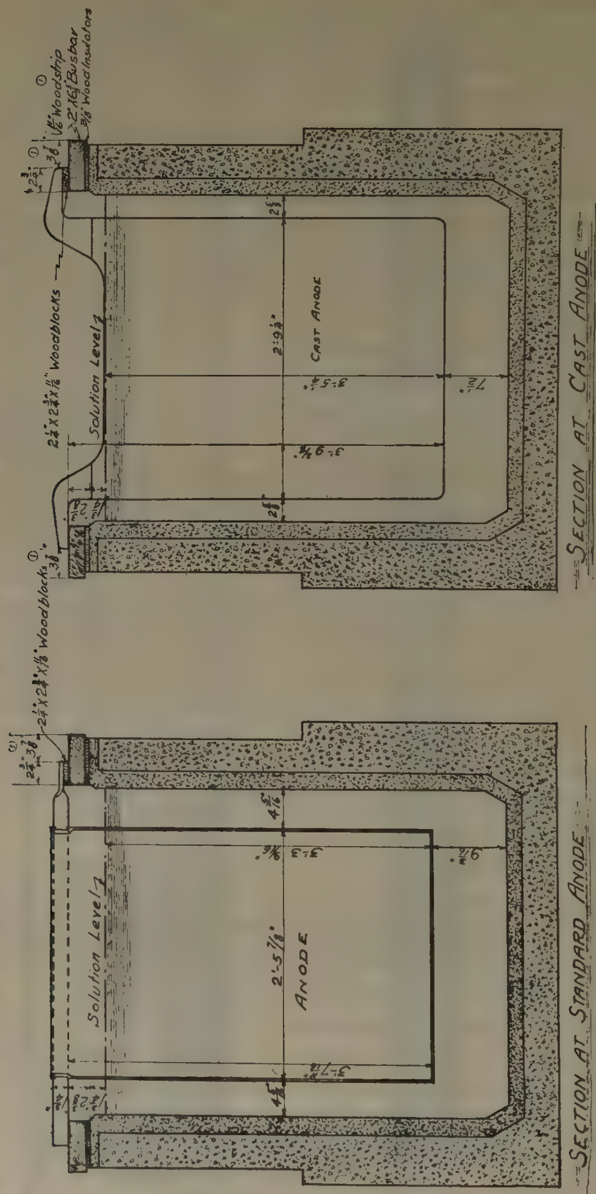
Section at Cast Anode
Cast Anode
Solution Level
Woodblocks
Section at Standard Anode
Anode
Solution Level

Fig. 11.—(Part 2)

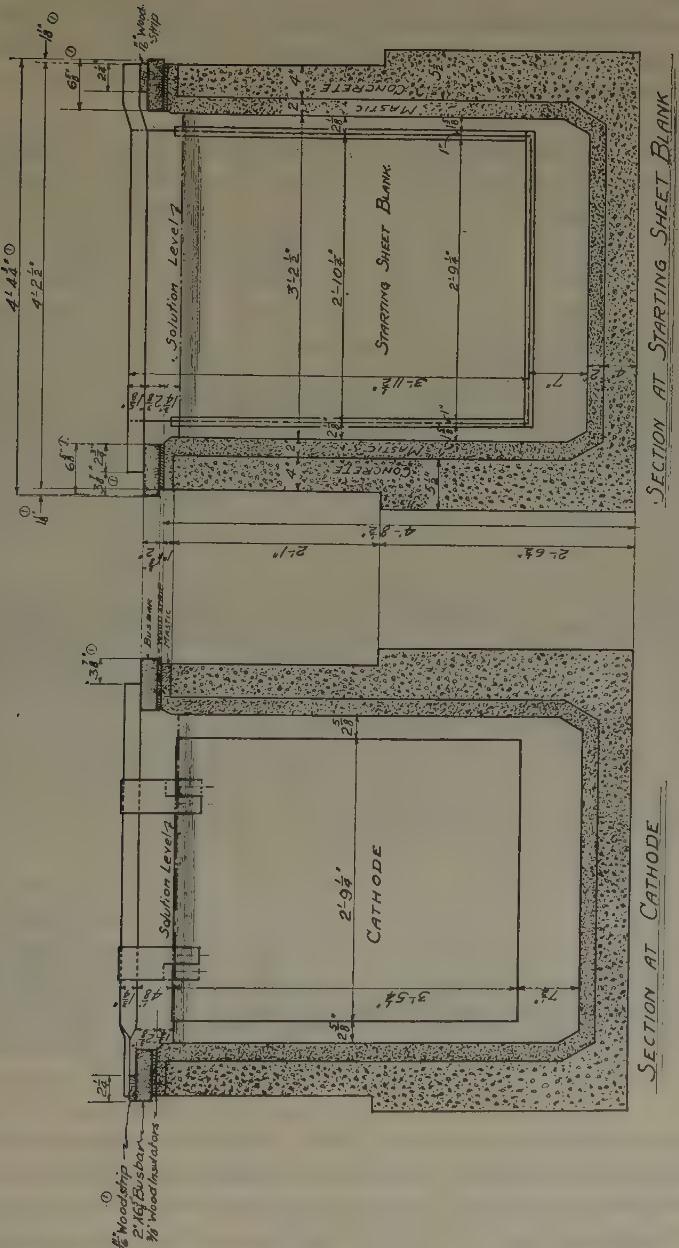

FIG. 11.—(PART 3)

FIG. 11.—GENERAL ARRANGEMENT OF ELECTROLYTIC TANKS.

compartment weir box and launder systems. Launders throughout the plant are of wood, lined with 6-lb. or 8-lb. chemical lead sheet, and where necessary are provided with renewable wood liners to protect the lead. Tailings are laundered to a gully below the plant site, which leads to a tailings pond constructed to impound them. Fig. 10 shows thickener section under construction.

Two small pump houses of brick and steel construction are provided near the thickeners, one to pump the solution from the last storage tank to the electrolytic tank house, the other to pump the solution for purification from the first storage tank to the purification agitators. Direct-connected electrically driven duriron centrifugal pumps are used, 8 by 6 in. for the first service and 6 by 4 in. for the latter. The sumps in the thickener tunnels and the agitator basement are also equipped with 6 by 4-in. duriron centrifugal pumps. All solution piping under high head is of steel, lead lined, and low-head piping is of extruded chemical lead. The sheet lead and pipe for the plant were fabricated in the company's own rolling and extruding mill, erected for the purpose. The pig lead was shipped in from the Broken Hill smelter in Rhodesia or from Europe.

Electrolytic Plant

The electrolytic tank house is a steel-frame structure, 266 ft. 6 in. by 405 ft. 0 in. in plan, sheathed with asbestos-protected metal and roofed with a wood deck covered with built-up roofing. The building is arranged with four crane aisles with a crane transfer at one end, to provide means for shifting cranes from one aisle to the other. The main operating floor is all on one level, and a basement is provided with ample head room for tank inspection and launder installation. There are 160 electrolytic tanks in the building, each 62 ft. 6 in. long, 3 ft. 2½ in. wide and 4 ft. 2½ in. deep inside (Fig. 11). The tanks are of reinforced concrete lined with 2 in. of asphalt mastic, and are equipped with a feed pipe at one end, an overflow dam at the other, and a hard lead plug and seat in the bottom for draining and cleaning out. The solution pumped from the leaching division is split into two equal parts by a weir tank in which the sulfuric acid requirement of the process is added continuously. The solution from the weir compartments is laundered to two constant-head tanks from which hard rubber header and riser pipes distribute the feed to the individual tanks. The overflow solution from the tanks is collected by a launder system in the basement and laundered through a tunnel to the agitator section of the leaching division. Conductor bars 2 by 6⅝ in. in cross-section are supported on wood insulating strips on the tank walls. The electrodes are arranged in three separate electrical groups in each tank, the groups being in series and the electrodes in multiple. The entire tank house is laid out in

two main electrical circuits, each taking normally 8000 amp. at about 460 volts. The center point of each electrical system is grounded through a low resistance in order to protect the workmen from injury. Sixteen of the electrolytic tanks are equipped so that they can be fed with a


12


13

Fig. 12.—Electrolytic tank house under construction.
Fig. 13.—Tank-house interior under construction.

separate circulation of pure solution, and enough of these tanks are so used to produce the necessary starting sheets from soluble anodes. The remainder of the tanks are equipped with insoluble anodes of hard lead. Electrodes are handled to and from the tanks by 5-ton tank-house cranes of the stiff-leg type. Forty-eight-hour starting sheets are stripped on racks at the end of the building, looped on standard Morrow clip machines, flapped on tables and suspended in racks ready for charging by the crane. Finished cathodes, 10 to 14 days old, are drawn from the tanks by the

crane, spray-washed and unloaded on cathode loading machines of the Great Falls type, which deposit them in piles on cars going to the furnace refinery. Figs. 12 and 13 show the electrolytic tank house under construction.

Furnace Refinery

The furnace refinery building is a steel-frame structure, 124 ft. 8 in. by 234 ft. 0 in. in plan, sheathed and roofed with corrugated galvanized steel, and arranged with two operating levels. The building is equipped with two 130-ton refining furnaces, each provided with a 38-ft. dia. casting wheel of the Clark type, together with bosh and inspection conveyors. The charging bay is served by a 7000-lb. charging crane, made by the Wellman Engineering Co., and the casting bay by a standard 5-ton crane. An outside runway equipped with a standard 5-ton crane provides handling facilities over the storage and shipping yard. The furnaces were originally designed for grate firing, but on installation were equipped for pulverized coal firing with Heyl and Patterson burners and feeders. The furnaces are constructed with air-cooled brick bottoms; each has its individual stack, but no waste-heat boiler. A small coal-pulverizing plant is next to the furnace refinery. Wankie coal from Rhodesia is dried in a rotary shell dryer, pulverized in conical air-swept ball mills and delivered to the furnace bins by screw conveyor.

Acid Supply and Electric Power

Sulfuric acid is supplied for the leaching from a plant built close to the thickener section by a subsidiary company of the Union Minière. It is a contact process plant, burning sulfur shipped in from the Texas fields and utilizing as a catalyst platinized silica gel.

The electrical substation, which distributes power to the entire plant, is a steel-frame structure with brick walls and a concrete slab roof covered with built-up roofing. The main building is 39 ft. 4 in. by 175 ft. 0 in. in plan, and houses the following equipment:

One main switchboard, 31 panels, remote control.

Three 4000-kw. motor-generator sets for electrolysis, each consisting of a synchronous motor driving two direct-current generators, each capable of delivering 4000 amp. at 500 volts. One of these sets is a spare.

Three 175-kw. motor-generator sets for supplying 250 volts direct-current excitation and auxiliary power. One of these sets is a spare.

Two air compressors with direct-connected synchronous motor drive, capable of delivering 3650 cu. ft. of free air per min. at 4600-ft. altitude and a pressure of 40 lb. These units, one of which is a spare, provide air for solution agitation, etc.

A lean-to on one side of the building, 28 ft. 3 in. by 122 ft. 6 in. in plan, houses the high-tension switches, auxiliary power transformers,

air washers for building ventilation and a storage-battery room. A basement is provided under part of the main building and the high-tension switch compartment, and houses the direct current breakers, main direct current buses, and switchboard auxiliaries. Power is supplied to the substation from the Panda steam plant or the Lufira hydroelectric plant at three-phase, 50 cycles, 6600 volts. The electrical equipment was largely supplied by the International General Electric Co. and the compressors by the Ingersoll-Rand Company.

Besides the major departments described above, the plant is equipped with its own general offices, mechanical shop, carpenter shop, warehouses, locomotive shop, laboratory, etc., etc. The ensemble provides a complete self-contained unit which turns out marketable copper from original ore. Fig. 14 shows a panorama of the plant.

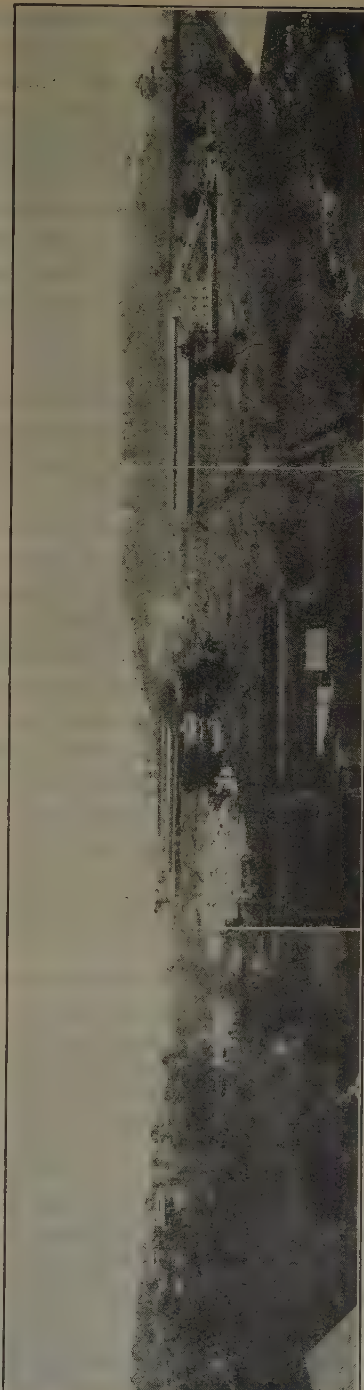

Fig. 14.—Panorama of 30,000-ton Panda leaching plant.

Details of Operation

The first unit of the new plant was ready to be started up in April, 1929, and was put into operation with a locally organized crew. As fast as additional units were completed, they were put into operation, and by the end of the year the entire plant was operating, except for four of the purification agitators, which still lacked some of their interior fittings. The plant was staffed with men whose training had practically all been obtained in the experimental leaching plant, and the operating crew was composed of men who had little if any experience with the type of equipment installed. The inexperience of the

operating force and the fact that construction was being carried on side by side with operation were the principal difficulties to be surmounted in starting operations. Mr. Eagle was sent into Katanga as the consulting engineer's representative to act in an advisory capacity during the final construction and starting up of the plant.

In general, the equipment installed gave excellent service. One major difficulty was encountered with the classifier mechanisms. The manufacturers supplied a mechanism which was not sufficiently strong to take care of the extra weight added by the lead covering specified for acid-proofing, therefore mechanical breakage occurred when the units were placed in operation. After new and stronger mechanisms were installed, there were no further difficulties. The screw conveyors in the drying and grinding division did their work satisfactorily, but maintenance of the intermediate bearings was excessive, and indicated the desirability of shifting the thrust bearing to the discharge end. The slope of the sand-tailings launder needed to be increased, because when the plant capacity rose nearly to its normal, the tendency of this clean material, free of slimes, to drop out of solution caused the launder as installed to overflow. The wood interior fittings of the Pachuca agitators were made too light to give long life and were replaced by heavier construction. The long electrolytic tanks, while ideal for use with insoluble anodes, resulted in such excessive solution segregation in the stripper tanks using soluble anodes that poor starting sheets were made. It was found necessary to insert baffles between the three electrical groups in these tanks to break up this segregation, and after this was done no further trouble was experienced. Except for the items mentioned above, practically no trouble of any moment was experienced with the equipment as installed.

TABLE 4.—*Summary of Plant Operations for December*, 1929

CRUSHING, STORAGE, DRYING, ETC.	
Wet ore crushed, metric tons, total	40,056
Moisture in wet ore crushed, avg. per cent	13.20
Cu in dry ore crushed, avg. per cent	6.57
Wet ore to driers, metric tons, total	39,984
Moisture in drier discharge, avg. per cent	0.24
LEACHING DIVISION	
Dry ore fed to leaching, metric tons, total	32,631.109
Dry concentrates fed to purification, metric tons, total	1,057.966
Cu in ore fed to leaching, avg. per cent	6.539
Cu in concentrates fed to purification, avg. per cent	28.700
Cu in classifier sands (per cent of dry solids), avg. per cent	0.442
Solution in classifier sands, avg. per cent	29.19
Cu in entrained solution in classifier sands, avg. grams per liter	11.84
Free H_2SO_4 in entrained solution in classifier sands, avg. grams per liter	0.00

TABLE 4.—(*Continued*)

Cu in acid slime tails (per cent of dry solids), avg. per cent	0.177
Solution in acid slime tails, avg. per cent	49.39
Cu in entrained solution of acid slime tails, avg. grams per liter	9.52
Free H_2SO_4 in entrained solution of acid slime tails, avg. grams per liter	0.00
Cu in purification slime tails (per cent of dry solids), avg. per cent	0.426
Solution in purification slime tails, avg. per cent	57.20
Cu in entrained solution of purification slime tails, avg. grams per liter	11.33
Free H_2SO_4 in entrained solution of purification slime tails, avg. grams per liter	0.00
Extraction (copper into solution), avg. per cent	96.53
Recovery (copper to tank house), avg. per cent	88.34
H_2SO_4 consumed per kilogram Cu recovered, kg	0.696
Sand tails, per cent of total tails	40.54
Acid slime tails, per cent of total tails	56.08
Purification slime tails, per cent of total tails	3.38
Metric tons total tails from 1 m.t. ore	0.869
Metric tons total tails from 1 m.t. concentrates	0.491
ELECTROLYTIC DIVISION	
Fine copper in cathodes drawn, metric tons, total	2,486.097
Cu produced per kw-hr., avg. kg	0.471
Ampere efficiency, avg. per cent	78.53
Amperes for month per circuit, avg	7,460
Volts for month per circuit, avg	444.69
Volts per electrical group (commercial), avg	1.97
Volts per electrical group (stripper), avg	0.39
Starting sheets stripped, total number	64,176
Good sheets, per cent	92.43
Starting sheet tanks in service, total number	12
Commercial tanks in service, total number	148
Tanks in service, total number	160
FURNACE REFINERY	
Furnace charges, total number	13
Hours per charge, avg	29.15
Metric tons to furnace per charge, avg	100.383
Metric tons production per charge, avg	86.454
Metric tons treated, total number	1,304.977
Metric tons produced, total number	1,123.896
Metric tons Wankie coal per charge, avg	24.905
SUBSTATION	
Alternating current power for deposition, kw-hr	5,458,000
Alternating current auxiliary power, kw-hr	876,000
Alternating current power, total, kw-hr	6,334,000

No process troubles were encountered, except in the furnace refinery, where the troubles were entirely the reflection of the inexperience of the operating force and the fact that the laboratory technique had not been developed to the degree of accuracy required by this rather delicate

operation. The matter of turning out a furnace refinery product that will meet the requirements of the fabricators is a difficult operation at the best, and the company exhibited a great deal of courage in attempting the job with a crew of the limited experience of the men they had.

TABLE 5.—*Typical Solution Assays and Screen Analyses*

AVERAGE SOLUTION ASSAYS

	Tank House		Acid Agitation Discharge, Grams per Liter	Purification	
	Feed, Grams per Liter	Discharge, Grams per Liter		Feed, Grams per Liter	Discharge, Grams per Liter
Cu	30.51	16.26	30.53	30.17	44.98
Free H_2SO_4	5.50	34.35	12.22	11.17	0.00
Fe, Total	5.63	5.63	6.65	5.81	1.99
Fe, Ferric	5.43	3.97	6.34	4.98	1.42
Fe, Ferrous	0.20	1.66	0.31	0.83	0.57
Al_2O_3	11.00	11.00	11.70	10.88	7.50

OTHER IMPURITIES ACCUMULATED IN THE SOLUTIONS, GRAMS PER LITER

Co	Mn	MgO	SiO_2	P_2O_5	CaO
10.03	6.50	13.60	1.72	3.80	0.34

AVERAGE SCREEN ANALYSES FOR DECEMBER, CUMULATIVE PERCENTAGE

Mesh	Ore to Leaching	Sand Tails	Slime Tails
+ 20	8.05	14.07	—
+ 35	22.95	42.09	—
+ 48	29.70	54.60	—
+ 65	35.95	66.98	—
+100	42.30	75.79	0.73
+150	47.25	82.07	2.08
+200	51.55	85.77	5.84
Total	100.00	100.00	100.00

Conditions in the copper industry had become so serious by the end of December, 1929, that the output of the new leaching plant was ordered curtailed in January. As a result of this, the largest month's production during the observation of the operations by the consulting engineers was obtained in December, with the plant still partly incomplete. A summary of the plant operations for that month is given in Table 4. Typical solution assays covering the principal constituents of the solution at various points in the process cycle are given in Table 5.

At the conclusion of the period of observation of the operations of the new plant at the end of January, 1930, it had been brought up to a reasonable degree of efficiency, but had still a considerable way to go to get on what would be considered a proper operating basis. The plant, however, had amply demonstrated its ability to meet without undue difficulty the promises made as to capacity, copper extraction and recovery, acid consumption, power consumption and over-all operating costs. A rather satisfying complement to these results was the fact that the final records of construction cost indicated that the plant had been built within the original estimate submitted in 1923.

DISCUSSION

(*Francis R. Pyne presiding*)

F. R. Pyne, Perth Amboy, N. J.—The authors mention (p. 620) the corrosive leaching solution. Was that due to ferric iron or something else?

H. Y. Eagle.—We do not know. The chemical composition of the solution is not sufficient, as far as we can see, to explain the extremely corrosive nature of the solution. We had various bronze and brass pumps, for example, in use. Inside of two weeks the pumps were like sieves. We could drive copper nails below the solution line in the agitators and the next day they were gone. I do not believe the amount of ferric iron in the solution was sufficient to account for it.

F. R. Pyne.—The amount of ferric iron appeared to be rather small to have such a powerful corrosive effect. There could not be nitrates very well.

H. Y. Eagle.—No nitrates and no chlorides. The presence of cobalt might account for it, but we do not know. The solutions were very much more corrosive than ordinary refinery solutions; the amount of cobalt in the solution somewhere in the neighborhood of 10 grams per liter was the only thing that to us seemed to be unusual.

Member.—To what extent do the authors think leaching may supersede smelting of the Katanga ores, or what proportion of the ores is likely to be leached and what proportion smelted?

H. Y. Eagle.—That is pretty much a matter of company policy, but our own opinion is that from the standpoint of costs the more of the ores amenable to leaching, which are leached, the more profitable it will be. Up to date, however, the amount of production presented by leaching is comparatively small. I think that in 1930 of the total production of about 139,000 metric tons only 21,000 of it was from leaching.

E. L. Jorgensen, Irvington, N. J.—Why was it decided to treat relatively coarse material by agitation in preference to separating coarse and fines, using percolation for the coarser material, and the method you are using for the finer material?

H. Y. Eagle.—We did considerable figuring on that basis and the conclusion we came to was that it is all well and good to separate the coarse material, and it is quite possible to leach the coarse Katanga material by percolation. There is no doubt that percolation leaching is a good deal more efficient metallurgically than is agitation, but when designing a slime plant the size of the slime plant is determined by the amount of very fine material in the ore. That is, the amount of settling area that must be provided to take care of the slimes is the same whether the coarse material is taken out or not, or practically so. So it meant that if we put in a coarse percolation

treatment for the coarse, we still had to build just as big a plant for the slimes, and it ran into too much money.

F. R. Pyne.—Is the source of the Katanga power water or fuel?

H. Y. Eagle.—The new leaching plant was started on the basis of steam power. I left Katanga in February, 1930, and in May of that year the new Lufira power plant was scheduled to deliver hydropower to the leaching plant; I suppose that is now in operation.

A. Grönningsater, Sudbury, Ont.—I believe that is due to the ferric sulfate. I have had experience with that kind of solution, in which I know there was no impurity except a small amount of iron. Only one or two grams per liter will eat up copper. If you take out the iron and leave everything else in, copper will withstand corrosion for years.

H. Y. Eagle.—The small amount of ferric iron in our solution has not seemed to us to explain the cause. In refinery practice there may be quite a bit of iron in the solution, and usually it splits up about fifty-fifty; that is, about 50 per cent ferric and 50 per cent ferrous. Often the amount of ferric iron in a refinery solution will greatly exceed the amount we had and still we never noticed that particularly corrosive action there. We were looking for the cause, but never hit upon a satisfactory explanation.

I would like to speak of our long electrolytic tanks. The long electrolytic tank, when it was presented at a meeting of the Electrochemical Society was looked upon more or less as a fish story. The energy efficiencies as high as 1⅛ lb. per kilowatt hour, which we obtained, were a good deal higher than the usual practice. The tanks, as they are now operating, while they are not yielding as much as that, owing to the low ampere efficiency produced by a new crew, will, I believe, yield a considerably higher figure eventually.

F. R. Pyne.—Has it been your experience that lengthening the tank increases the power efficiency?

H. Y. Eagle.—Yes.

F. R. Pyne.—I should think it would because there is the same solution throughout no matter how long the tank is. And a high acid can be run whereas in a short tank the same beneficial results are not obtained.

H. Y. Eagle.—I should like to know whether in any other industry, for example in electrolytic zinc, or other copper plants, the same thing has been experienced.

F. R. Pyne.—We have not tried long tanks for copper—I have never seen any long tanks—but we have experienced the thing to which you refer in the paper; that is, that when the circulation in the tank is slow, it is practically constant from one end of the tank to the other, except possibly where the solution comes into it. I presume you find the same condition in the new longer tanks.

H. Y. Eagle.—It persists up to 60 ft., as far as we have tried it.

W. C. Smith, New York, N. Y.—What experience have you had with insoluble anodes?

H. Y. Eagle.—The insoluble anodes that we first used in the experimental plant were made of chemical lead, and naturally we ran into a lot of trouble. The warping that developed after a short time was serious, and we had to take the anodes out of the tank repeatedly and beat them straight. The oxidation was very serious and the chemical lead anodes did not last very long. We tried out lead anodes of various antimony contents, and finally fixed on one with a content of about 6 per cent anti-

mony, that being a point that gave practically no shrinkage in the mold and seemed to be as resistant to warping and corrosion as any of the others tried.

W. C. Smith.—Have you had any experience with the silver-lead or perforated anodes?

H. Y. Eagle.—I have tried the perforated anodes but not the silver-lead. We tried the perforated anodes on the basis of some tests which I think were run at Raritan. Perhaps Mr. Pyne can confirm this. Was it not at one time standard practice at Raritan to operate the liberator tanks with a grid anode?

F. R. Pyne.—Yes, some years ago, but it has not been done since I have been there. It was abandoned so I should imagine it was unsatisfactory. Whether it was due to the poor casting of the anodes, or some other defect inherent in the shape of the anodes, I cannot say.

H. Y. Eagle.—It was generally considered at Raritan, I think, that the same voltage was obtained for a given liberator tank, irrespective of the anode density, and it was a considerable saving in lead to make it a grid. In a leaching plant, of course, with all insoluble anodes, the saving in lead would be enormous if a quarter of the weight could be cut out. We tried it, but the voltage ran up perceptibly with added anode density. Apparently tank voltage is dependent on both cathode and anode density. At least, that was our experience.

A. Grönningsater.—The new copper refinery at Sudbury is using a grid anode. There is said to be no increase in voltage. How were your anodes cast, and how thick were they?

H. Y. Eagle.—They were cast in a cast-iron chill, and if I remember rightly were 3/8 in. thick. At the expiration of my stay at Katanga, when they had been in about eight months, there seemed to be no perceptible depreciation. I have an idea we may get anywhere up to 10 years out of them.

About ten amperes current density is being used. The design of the plant was on the basis of 100 amp. per square meter.

E. L. Jorgensen (written discussion).—When I was in Chile with the Braden Copper Co., we carried out in 1914 and 1915 extensive experiments under operating conditions. We compared grid anodes cast in a similar manner to that which you have described with sheet-lead anodes. When starting with new anodes with clean surfaces, the sheet-lead anodes showed 5 per cent lower voltage than the grids, with current densities at the cathode of 10 to 15 amp. per sq. ft. With 15 to 40 amp. per sq. ft. this advantage was lost. After a very short period of service, the voltage increased rapidly 75 to 80 per cent with sheet-lead anodes, whereas the grids showed no increase at all after 70 days of service. The saving in power with grids was over 40 per cent.

Based on 70 days operation, we estimated that the grids would not have to be remelted oftener than every four years, with the possibility of much longer life.

Another important observation was that we could make good starting sheets with grid anodes but not with sheet-lead anodes. The reason for this was that the evolution of oxygen was concentrated on a small surface and acted as an air-lift to promote good circulation where it was most needed. The cathodes and anodes were spaced 5 in. on centers.

The findings given are so much at variance with the observations of others that some details should be given. The grids were made of 6 per cent antimonial lead and the others of chemical sheet lead. The current density on the grids was 3.4 times higher than on the sheet-lead anodes. The electrolyte had the following composition in the experimental tanks: copper, 17 to 20 grams per liter; ferrous iron, 20 to 26;

ferric iron, 11.5; sulfuric acid, 45 to 70; nitric acid, none; chlorine, none. The temperature was 50° to 55° C.

Since I am talking of the Braden Copper Co., a few remarks on leaching operations there might not be out of order. We roasted sulfide concentrate and leached with sulfuric acid solutions. We removed ferric iron from neutralized solutions as basic sulfate and separated the precipitate from partially leached pulp by classification, as you did later at Katanga. This was a relatively simple matter when working with concentrate from Wilfley tables. With the much finer flotation concentrate this classification was not so good. The difficulty was overcome, experimentally at least, with magnetic separators covered with lead. The basic iron sulfate was nonmagnetic and partially leached calcine was strongly magnetic.

C. W. Eichrodt, Laurel Hill, N. Y.—I am not with Chile Copper Co. now, but in 1924 I presented a paper on the electrolytic tank house for that company at the same time that Mr. Eagle did. At that time we had experimented with larger tanks, but failed to mention the matter. Our experience in this connection was that we lost in ampere efficiency more than we gained in reduced voltage. Our development along these lines was somewhat a matter of secondary consideration. In order to improve the flow of electrolyte between tanks or cells of our electrolytic sections, which consisted of 16 tanks in series, a new type of construction was developed. The section was built as one large tank with partitions that divided it into the individual cells; openings in the partitions were provided for the flow of electrolyte through the series. The size of these openings was increased until the final development was a tank containing 16 electrode sets in series with partitions between sets consisting only of mastic-covered supporting columns for the beams carrying the intermediate transfer bars. The voltage was lower, mainly, I believe, because of irregular current distribution, and the ampere efficiency loss for the same reason was lowered to the point that the net power efficiency decreased.

H. Y. Eagle.—Did you have baffles between the various tanks?

C. W. Eichrodt.—No, in the one tank mentioned we did not. Any suitable material for baffles would not stand up in our electrolyte. We made little mention of our development and immediately replaced portions of the mastic partitions, which might be considered baffles.

H. Y. Eagle.—We found that in order to cut down the leakage between the various electrical groups in the tank it was essential to put at the end of each electrical group one anode or one sheet of lead, which was absolutely not in circuit. That meant that for current to leak from one group to the next, there would necessarily be two voltages of decomposition. This lead screen absolutely stopped the leakage from one group to the other and kept ampere efficiency up.

C. W. Eichrodt.—Our individual electrode sets consisted of 56 anodes and 55 cathodes, with an anode at each end. This might be considered a long tank unit in itself. It was the partitions between the sets in series that we removed. No screen or baffle between sets was practical because no inexpensive material that was suitable would stand up in our solutions, which contained nitric, hydrochloric and sulfuric acids and ferric iron. With the improved alloys now available, something might be developed, but I doubt whether it would prove practical.

You mentioned that you did not have much success with acid-resisting materials. I am inclined to believe that cobalt and not ferric iron is basically responsible. Ferric iron behaves in no consistent manner. I have seen four or five grams per liter apparently do no harm, and again under practically the same conditions one gram per liter cause trouble. It depends upon the agents causing alternate oxidation and reduction

of the iron. If the solution is saturated with air, trouble from iron may be aggravated. In this connection, the presence of other metals even in relatively negligible quantities is important. There is apparently what might be called a catalytic action in speeding up the oxidation and reduction of iron, exerted by certain metals, one of which I am inclined to believe is cobalt. The cobalt would be the primary source of trouble while the iron takes part in the corrosion reaction.

F. R. Pyne.—When talking with Mr. Wheeler some years ago about the Katanga property I understood that there is either a scarcity of wood or that it is all dwarf wood. What are you using for poling in the furnace refining?

H. Y. Eagle.—There is no scarcity of wood in Katanga. There is an abundance of what is called "bush," and it is not dwarf wood by any means, but an extremely gnarled wood. It is not suitable for construction but for poling it is quite all right.

F. R. Pyne.—Is it a hard wood?

H. Y. Eagle.—It is a bastard mahogany, mostly.

C. W. Eichrodt.—How much fresh water for washing is added to the leaching-solution circuit per ton of ore leached?

H. Y. Eagle.—I cannot give you that in definite figures, but I can say that the amount of wash water added in the classifiers is one volume for each volume of entrained solution in the classifier tails. These tails go out at about 30 per cent moisture. The same figure applies to the amount of wash water added in the slime treatment, one volume for each volume of entrained solution in the slime tails, and the slime tails average about 50 per cent moisture. From the figures in the paper of the relative amounts of sands and slimes, and the relative moisture contents of the two different kinds of tails, you can figure out for yourselves the amount of wash water per ton of ore. Does that answer your question?

C. W. Eichrodt.—Not exactly. The amount of moisture in the tailings must be equal to the water added, or the circuit will build up in volume unless there is an aid in the way of evaporation or deliberate discard.

H. Y. Eagle.—There is no discard in the system.

C. W. Eichrodt.—The water used in washing ore is equivalent to the tailings' moisture?

H. Y. Eagle.—Absolutely.

B. H. Strom, New York, N. Y.—I have had some experience with the effect of chlorine in the electrolysis of zinc sulfate solution, using anodes of soft lead. Above one-tenth gram per liter chlorine the corrosive action became noticeable; with one-half gram per liter the anodes became perforated in a few days. Many other factors influence anode corrosion in zinc electrolysis, and chlorine does not tell the whole story. I know of one case where the solution generally runs as high as 0.08 grams per liter chlorine. The anodes last for years; in fact, I have been informed that they could easily be made to last for eight years, if other conditions did not make a change advisable.

Coming back to the leaching plant: I noticed a reference to a patented device used to prevent segregation and plugging of the Pachuca tanks. No description is given, and I would like to have a little more information on the subject if possible.

H. Y. Eagle.—I do not know whether I can do it without a sketch, but I will try. When the material is lifted from the bottom of the agitator to the surface, naturally the coarse drops out before the solution gets over to the outflow of the tank, and the

result is that a comparatively small amount of coarse will go out from the ordinary type of agitator, either a Dorr or Pachuca.

We found that out after digging the system out a number of times, and came to the conclusion that our 35-mesh material simply could not be handled in a Dorr agitator with 2½ to 1 density. So we tried out various schemes, and finally came to one, which in substance is this: We take all of the discharge of the central air-lift, put it into a launder, take that launder directly to the outer part of the tank. Besides the outlet port, there is at that point a very much larger port, which permits the solution to run back into the tank. The size of the exit port is such that it will discharge the amount of feed to the tank with a comparatively small head, and the port is equipped with a constricting gate, which provides for any possible and necessary adjustment. When the tank is started, and there is the condition of a given amount of inflowing pulp and a certain rate of circulation of the central air-lift, you adjust the constricting gate on the exit port from the tank until the level in the tank will maintain itself just about at the point that will smother the weir of the discharge from the collecting launder back into the tank. When this is done, the tank will automatically maintain its level with comparatively little attention, the reason for it being that if more solution goes out of the tank from the exit port than comes in, the solution level in the agitator will fall. As soon as it falls it takes the smother off the weir. That automatically increases the discharge of the weir and brings back the level again. If less goes out of the tank than is coming in, the solution level in the tank rises, raises the head on the outlet port and automatically increases the discharge from the exit port. Since all the coarse and fine material in the internal circulation is given an equal chance to leave the tank, the amounts of these discharged are in the same proportion as they exist in the feed.

The device actually works and we are handling 1600 tons daily without trouble. Often when there is a breakage of copper screens, we get considerable oversize in the bins, and even up to as high as ½ in. it is handled successfully.

E. L. Jorgensen.—How are the extractions in the ½-in. material?

H. Y. Eagle.—Not good. The extractions we get under the average grinding conditions are given in the paper, and naturally in the coarse material the extractions run downward.

E. L. Jorgensen.—It would improve if you gave it time?

H. Y. Eagle.—Yes. On the other hand, I would not recommend anybody to try to agitate stuff that is ground through ½ inch.

E. L. Jorgensen.—In Chile, we leached coarser material by percolation for an extended period of time. Have you any conclusions for Katanga ore on this point?

H. Y. Eagle.—I hope you understand that we are not presenting this process as an argument that agitation is better than percolation. We would have been pleased to leach on a basis of percolation if we could have done so, but Katanga ores are not amenable to percolation. Our extraction and recovery on 6.5 per cent ore looks like very poor work, but it is the best we can do under the existing conditions.

E. L. Jorgensen.—What characteristic of Katanga ore makes it unsuitable for percolation?

H. Y. Eagle.—In the first place, the mineral in the Katanga ores is largely malachite, which is a carbonate. You know what happens when you put acid on a carbonate. The upper percolation will not work because the tremendous gas evolution takes every bit of the fines out of the tank, and the downward percolation will not work for the same reason. But even were it not for that, the Katanga ores are largely malachite

in shales and sandstones, and the amount of colloidal clayey material is great enough so that the solution will not percolate through it on a downward basis.

MEMBER.—I should like to ask the gentleman from the Chile Copper Co. about the use of antimonial lead anodes containing a high percentage of arsenic, say 2 per cent arsenic, or over.

C. W. EICHRODT.—While I was there we did not use antimonial lead anodes except in laboratory experiments. I do not believe that we ever tried arsenic in the anodes.

MEMBER.—I am not asking the question from the standpoint of an electrolytic copper man but rather from the standpoint of the producer of antimonial lead. I had heard that certain plants in South America were using antimonial lead high in arsenic.

F. R. PYNE.—I have never seen it used in copper refining, but I would like to hear some results from it if anybody has tried it out.

C. R. HAYWARD, Cambridge, Mass.—I know nothing definite regarding high-arsenic anodes, but in the early days of experimentation on the Chuquicamata process I examined a good many anodes, lead and otherwise, and it was rather interesting to note that those that were cast in a flat mold corroded only slightly on the side that was chilled; that is, the side that came in contact with the mold, whereas the upper side went to pieces rapidly.

A. B. YOUNG.—The plant referred to is the Potrerillos plant of the Andes Copper Co. The anodes were high in arsenic, 2 per cent or better. However, it is my understanding that the arsenic did not prove to be of any particular benefit.

R. C. DALZELL, Trenton, N. J.—I have always found pH control very helpful in an iron precipitation such as takes place in the purification section described in this paper. Neutrality covers a broad field, and I should like to know the approximate pH of the neutral thickener solution referred to on p. 623, third line from the bottom. I notice that the solution from purification contains 2 grams per liter of iron. I assume that this is ferrous iron, and that chemical oxidation of it is more expensive than would be justified. Air oxidation might be applicable here. A method that appeals to me because of its simplicity and cheapness consists in circulating the solution downward through a sort of chimney, in which are placed criss-crossed baffles. These break up the solution, presenting a very large surface for oxidation by the air drawn up through the chimney. The hot solution creates a natural draft in the chimney. The only operating expenses are power for circulating the solution and steam for heating the solution. The operation is very effective and has been in use for some years by a large European refinery.

The Inspiration Leaching Plant

By Harold W. Aldrich* and Walter G. Scott,† Inspiration, Arizona

The leaching process of the Inspiration Consolidated Copper Co. is based upon the solubility of oxidized copper in sulfuric acid and the solubility of sulfide copper, largely in the form of chalcocite, in ferric sulfate solution. This necessitates the carrying of a rather large amount of iron in solution. The ferric sulfate is regenerated from ferrous sulfate by the oxygen given off at the anode during electrolytic precipitation of the copper.

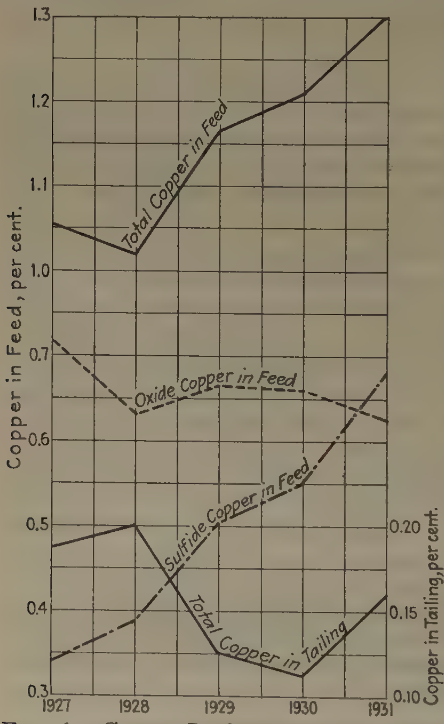

Fig. 1.—Copper in feed and tailing, Inspiration leaching plant.

In September, 1924, excavation was started at Inspiration, Arizona, for the construction of a plant to treat 7500 tons of ore per day. When the plant was begun, the fact developed that by a minor change in the height of the overflow on the leaching tanks, 9000 tons could be charged into each tank and treated satisfactorily. This determined the maximum capacity of the plant as 9000 tons per day. The first ore was charged in October, 1926, and there are now available the records of five complete years of operation; namely, 1927 to 1931, inclusive.

The original estimate of copper content of the ore to be mined for this plant was 1.19 per cent, of which 0.77 per cent was as oxide and 0.42 per cent as sulfide. From the metallurgical results obtained at the 35-ton pilot plant, it was estimated that a 90.0 per cent extraction of the oxide copper and a 75.0 per cent extraction of the sulfide copper could be made. It was estimated that the tailing would carry 0.182 per cent total copper, resulting in a

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† Assistant Superintendent of Leaching Plant, Inspiration Consolidated Copper Co.

total extraction of 84.7 per cent. The actual results of the first two years of operation were tailings assaying 0.188 per cent total copper for the first year and 0.200 per cent for the second year. It is interesting to see (Fig. 1) that the oxide copper content of the ore has remained uniform and that the increased percentage of total copper in the ore is caused by an increase in sulfide copper.

During the second year of operation, it was recognized that all types of ore within the limits of our mining operation were not physically suited to this process. Ore from certain locations in the mine carried an excessive amount of colloidal material, which seriously hindered solution percolation in the leaching tanks. The removal of primary slimes and the construction of a plant for their treatment were the major changes in plant construction. The flow sheet of the leaching plant is shown in Fig. 2.

Crushing Division

The coarse-crushing plant consists of gyratory crushers and horizontal-shaft Symons disk crushers, which reduce the ore from mine run to 1½ in. The crushed ore is delivered by belt conveyor from the coarse crusher to the ore bins at the leaching plant. The capacity of these bins is 10,000 tons, of which 8000 tons is available. The ore is drawn to a conveyor and thence to a bank of eight Hum-mer vibrating screens. The screen opening is varied from ⅛ by ¼ in. to ¼ by ½ in., according to the change in physical character of the ore. The undersize goes to the washing plant, where the slimes are removed, the sands going directly to the leaching tanks. The oversize from the screens goes to a pocket bin over rolls that operate as coarse rolls.

There are four sets of 78 by 24-in. face Traylor rolls having 9-in. shells. Two sets are operated as fine rolls and one set as a coarse roll, leaving one as a spare. The discharge from all rolls is delivered to 16 Hum-mer screens equipped with ⅜-in. Ton-cap screen cloths. Undersize from these joins the sand product from the washing plant and goes to the leaching tanks, while the oversize returns to pocket bins over the fine rolls. The coarse rolls are set at ⅝ in. and the fine at 5/16 in. The rolls operate at 110 r.p.m., and each set uses 275 hp. The method of shelling rolls has been changed several times, but the best results are obtained by shelling hot. The bearings are lubricated by a continuous oil flow, operating in closed circuit with an oil filter plant. Roll-shell data are given in Table 1.

In the operation of all crushing and conveying machinery, the drives are interlocked electrically, so that whenever a motor driving a conveyor kicks out, all machinery delivering ore back of it stops immediately.

The crushing of 9000 tons requires about 10 hr., and it is not uncommon to average better than 1100 tons of crushed ore per hour for several hours.

The conveyor delivering crushed ore to the leaching tanks is equipped with a weightometer. A bucket sampler cuts the discharge stream on the

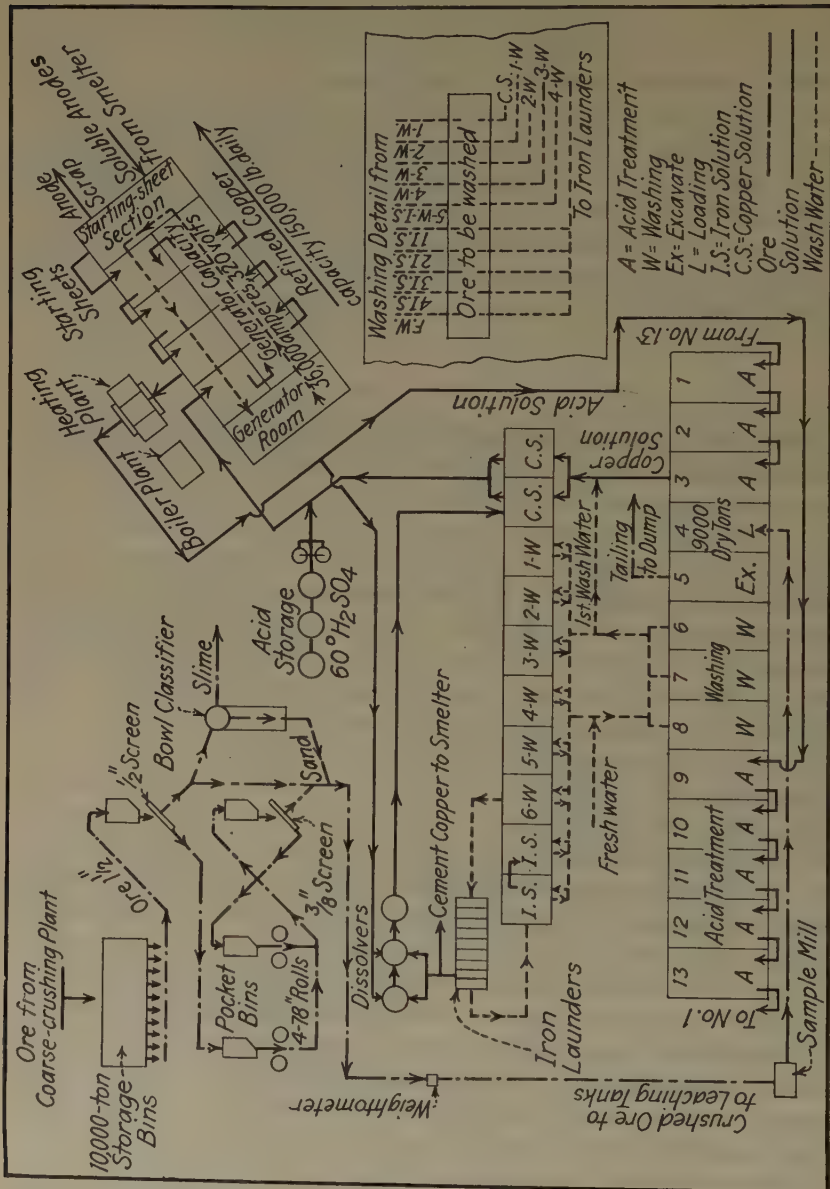

FIG. 2.—FLOW SHEET OF THE INSPIRATION LEACHING PLANT.

same belt, so that an accurate weight and an accurate sample of the ore in each charge are obtained. One conveyor travels the whole length of the bank of leaching tanks, a distance of 1000 feet.

TABLE 1.—*Roll-shell Data*

Date	Dry Tons Crushed	Pounds Steel per Ton Ore Crushed
9/27/26 to 12/31/27	3,469,394	0.06501
Year 1928	3,040,171	0.06427
1929	3,052,268	0.09159
1930	2,518,974	0.09804
1931	2,402,479	0.06938
Total to date	14,483,286	0.07693

Two separate cyclone dust-collector systems are operated, one in the fine-crushing plant (Fig. 3) and the other in the screening plant. These are identical in design, being equipped with Sturtevant 72-in. suction fans driven by 50-hp. motors. The suction ducts are connected to all units

FIG. 3.—FINE-CRUSHING PLANT AT INSPIRATION.

making dust, such as the rolls, screens, conveyors, etc. The rolls and screens are tightly housed, while the conveyors are enclosed at all points where ore is fed to them. The collected dust is delivered as a slime product to the bowl-classifier overflow in the washing plant.

WASHING PLANT

In the fall of 1928, plans were made for experimentation in removal of slimes from the ore. The separation was to follow the coarse crushing. Full-sized classifiers were installed, and, as there was no treatment plant

to handle the slime product, this material was stored for future treatment. The extraction on the charges so treated showed such improvement that thereafter the primary slimes were removed from all ore charged to the leaching tanks. Table 2 shows that in the year 1929 and succeeding

TABLE 2.—*Metallurgical Results*

	Year				
	1927	1928	1929	1930	1931
Feed					
Oxide copper, per cent	0.718	0.632	0.663	0.658	0.625
Sulfide copper, per cent	0.338	0.387	0.504	0.550	0.681
Total copper, per cent	1.056	1.019	1.167	1.208	1.306
Tailing					
Oxide copper, per cent	0.048	0.053	0.023	0.020	0.020
Water-soluble, per cent	0.011	0.012	0.001	Tr.	Tr.
Sulfide copper, per cent	0.129	0.135	0.103	0.092	0.142
Total copper, per cent	0.188	0.200	0.127	0.112	0.162
Extraction					
Oxide copper, per cent	91.78	89.72	96.38	96.96	96.80
Sulfide copper, per cent	61.83	65.12	79.56	83.27	79.15
Total copper, per cent	82.20	80.37	89.12	90.73	87.60

years the metallurgical practice was materially better than in 1927 and 1928. The oxide copper extraction since slime removal has remained fairly constant at 96.0 per cent, while the sulfide copper extraction has averaged 80.66 per cent.

The washing unit consists of two Dorr bowl classifiers. The bowls are 25 ft. in diameter and the arms revolve at 2 r.p.m. The rake compartment of each classifier has a total width of 16 ft. and contains four rakes, each 42 in. wide. The rake length is 33 ft. 4 in.; speed 26 strokes per minute; slope of bed 2 in. per foot. The bowl overflow runs to a sump, from which it is pumped by an 8-in. Wilfley sand pump through a 12-in. wood-stave pipe to the concentrator. The pulp is then subjected to flotation treatment for recovery of the sulfides, preceding the acid agitation treatment for recovery of oxide copper.

When first put into operation, the bowl classifier produced a wetter sand product than did the classifier used in experimental work. This was attributed to the bed slope, the reciprocating classifiers used in the experiment having 3 in. to the foot. To remedy this, a 3-ft. extension having a slope of 4½ in. to the foot was added to the rake compartment. Although this reduced the moisture content from approximately 24 per cent to 21 per cent, it also reduced the classifier capacity approximately one-third.

As will be discussed later, for the best metallurgical results, it is desirable that wash water be the only additional water added during this

process. The moisture content of the sands, therefore, has received considerable attention. Owing to the presence of ferrous sulfate, leaching solution cannot be used in the washing process on account of the succeeding operation of flotation on the slime product.

Assuming 1931 to be an average year, 5.75 per cent of the ore received from the mine was sent as classifier overflow to the concentrator and slime-leaching plant for further treatment. The classifier sands contained an average of 20.2 per cent moisture and 3.3 per cent minus 200-mesh material. Classifier feed averaged 24.9 per cent, and the overflow 83.4 per cent of minus 200-mesh material. Water used per ton of slime amounted to 1045 gal. Overflow results were as follows:

Dry tons overflow	146,468
Overflow oxide copper, per cent	1.156
Overflow sulfide copper, per cent	0.389
Overflow total copper, per cent	1.545
Pounds copper per dry ton overflow	30.90

Leaching Division

Fig. 4 is a general view of the Inspiration leaching plant, and Table 3 shows annual metallurgical reports for five years. The 13 concrete, lead-lined leaching tanks are each 175 ft. long, 67.5 ft. wide, and 18 ft. deep, having an approximate capacity of 9000 dry tons of ore per tank. They are lined with 7-lb. chemical lead. On the side walls the lead is protected by a covering of 2-in. planks held in place by vertical posts. The filter boards are of 2-in. material, having fifteen ⅜-in. holes per square foot of surface, each countersunk with a ¾-in. hole from beneath. The tank bottom slopes slightly toward the center as well as toward the end where the drain pipe is located. There is but one opening in the bottom of each tank, a 14-in. lead pipe burned to the lead lining, entering at the opposite end from the overflow, just above the tank bottom and below the filter bottom. All solution enters the tank through this pipe and all drainage is taken out through it.

The lead work in the leaching tanks has a tendency to buckle or creep, and often leaks are found at these places. This is not serious, however, as the sun never comes in direct contact with the lead lining, which is protected by the wood covering. The lead in open lead-lined launders constantly buckles, but because of their accessibility they can be repaired easily.

In charging, the ore is distributed uniformly over the entire tank. A standard tripper discharges the ore from the conveyor, which travels the length of the leaching tanks to a belt traveling the length of a movable spreading machine that spans the bank of tanks. An automatically reversing tripper travels on the spreader and empties the load of the belt into the tank. At each reversal of this tripper, the spreader bridge

TABLE 3.—*Annual Metallurgical Reports*, 1927–1931

	Previous to Removal of Primary Slimes		Primary Slimes Removed		
	1927	1928	1929	1930	1931
Heads					
Charge Nos. 51–751, 1386–1657					
Tank No.					
Ore charged, tons	3,099,639	3,182,546	3,300,467	2,713,686	2,582,473
Moisture, per cent	4.666	4.423	7.507	7.216	6.870
Ore charged, tons dry	2,955,014	3,041,334	3,052,692	2,517,855	2,405,057
Oxide copper, per cent	0.718	0.632	0.663	0.658	0.625
Sulfide copper, per cent	0.338	0.387	0.504	0.550	0.681
Total copper, per cent	1.056	1.019	1.164	1.208	1.306
Ore on 4 mesh, per cent	41.5	37.2	18.2	30.2	33.5
Ore through 20 mesh, per cent	25.3	30.7	31.1	26.5	24.8
Ore through 200 mesh, per cent			11.0	9.0	8.9
Tailings excavated					
Charge Nos. 51–751, 1386–1657					
Tank No.					
Moisture, per cent Heads	11.647	11.695	10.899	10.675	10.300
Oxide copper, acid-soluble, per cent	0.048	0.053	0.023	0.020	0.020
Oxide copper, water-soluble, per cent	0.011	0.012	0.001	Tr	Tr
Sulfide copper, per cent	0.129	0.135	0.103	0.092	0.142
Total copper, per cent	0.188	0.200	0.127	0.112	0.162
Oxide copper extraction, per cent	91.783	89.715	96.380	96.960	96.800
Sulfide copper extraction, per cent	61.834	65.116	79.563	83.273	79.148
Total copper extraction, per cent	82.197	80.373	89.117	90.728	87.596
Copper dissolved per ton of ore, lb	17.36	16.38	20.80	21.92	22.88
Solutions					
Ferric iron in solution from leaching tanks, grams per liter			3.2	2.2	2.8
Acid solution advance, gal. per min	1,427.6	1,761.5	1,947.5	1,848.1	2,319.3
Acid solution advance, total gal	750,367,000	928,367,456	1,023,592,000	971,383,000	1,219,001,560
Copper solution to tank house, gal. per min	1,570.5	1,836.0	1,947.5	1,848.1	2,319.3
Copper solution to tank house, total gal	825,470,000	967,646,608	1,023,592,000	971,383,000	1,219,001,560
Specific gravity of solution to tank house	1.192	1.165	1.171	1.181	1.188
Copper in solution to tank house, grams per liter	28.2	23.4	28.2	33.1	26.4
Copper in solution from tank house, grams per liter	22.3	18.2	23.1	26.6	21.6
Acid in solution to tank house, grams per liter	41.4	30.6	25.5[a]	26.9[a]	33.8[a]
Acid in solution from tank house, grams per liter	56.6	45.0	42.4[b]	38.4[b]	41.1[b]
Total iron in solution to tank house, grams per liter	17.3	16.0	18.6	16.4	20.0
Ferric iron in sol. to tank house, grams per liter	5.9	5.3	4.5	3.1	4.0
Ferric iron in sol. from tank house, grams per liter	10.2	9.3	10.0	9.1	8.6
Solution temperature to tank house, deg. C	32.4	31.8	32.6	30.7	34.1
Solution temperature from tank house, deg. C	42.2	40.0	41.6	41.5	40.7
Solution temperature from heating plant, deg. C	50.2	47.6	43.9	45.4	44.0
Atmospheric temperature, deg. C	17.7	18.7	18.9	19.1	19.3
Oil burned, gal	881,490	1,107,172	717,572	789,382	1,138,207

Tank house—commercial division					
Total weight commercial copper pulled, tons	47,833,425	42,707,143	50,568,452	48,423,922	49,245,579
Total weight cathodes pulled, tons	54,404,845	49,399,338	59,030,118	56,243,070	57,981,074
Average weight of cathodes pulled, pounds	100.4	91.6	86.3	88.3	80.8
Average voltage between anodes and cathodes	2.36	2.24	2.35	2.35	2.21
Average current density	14.2	11.8	14.6	13.3	11.9
Number of tanks on cathodes	117.2	120.0	120.0	119.3	117.0
Tank house—starting sheet division					
Number of tanks on starting sheets	13.0	14.7	15.6	16.9	17.5
Number of starting sheets made	662,542	647,311	810,234	799,406	862,200
Average weight of starting sheets, lb	10.6	11.3	11.3	11.2	11.4
Pounds soluble anodes received—Lot Nos. 19–172, 415–607	10,015,240	10,298,480	12,962,400	15,020,340	18,143,380
Pounds anode scrap shipped—Lot Nos. 1–104, 218B–310A	2,234,520	2,442,540	3,096,160	4,512,922	4,361,961
General					
Pounds electrolytic copper shipped—Lot Nos. 23–955, 2064–2607	55,632,371	50,588,749	60,278,162	58,198,306	59,090,589
Gross a.c., kw-hr. to sub-station	78,972,009	65,757,166	85,758,529	78,541,627	70,135,807
Gross d.c., kw-hr. to tank house	72,305,451	59,407,654	76,325,238	71,532,266	64,210,610
Conversion efficiency, per cent	92.1	92.0	92.0	92.0	92.0
A.c. per pound copper, kw-hr	1.641	1.540	1.695	1.615	1.431
Av. ampere efficiency on produced copper, per cent	61.43	64.66	61.75	64.05	67.93
Pounds 60° Bé. acid per ton ore	31.36	26.5	33.3	23.4	23.3
New water used for washing, gal	71,953,300	73,246,920	37,568,350	54,416,000	50,239,000
New water used for classifying, gal			281,374,585	186,528,000	153,064,000
Pounds copper as cement copper shipped—Lot Nos. 20–223, 578–684	4,079,503	7,321,914	13,372,609	7,114,210	5,528,773
Pounds copper as copper slimes shipped—Lot Nos. 1–16, 39–44	100,919	210,044	176,118	124,705	145,909
Commercial tank cleanings, Lot Nos. 3–5				123,966	211,508
Dirty starting-sheet scrap				160,717	

[a] Free acid, 17.0 grams per liter.
[b] Last six months of 1931, 24.5 grams per liter.

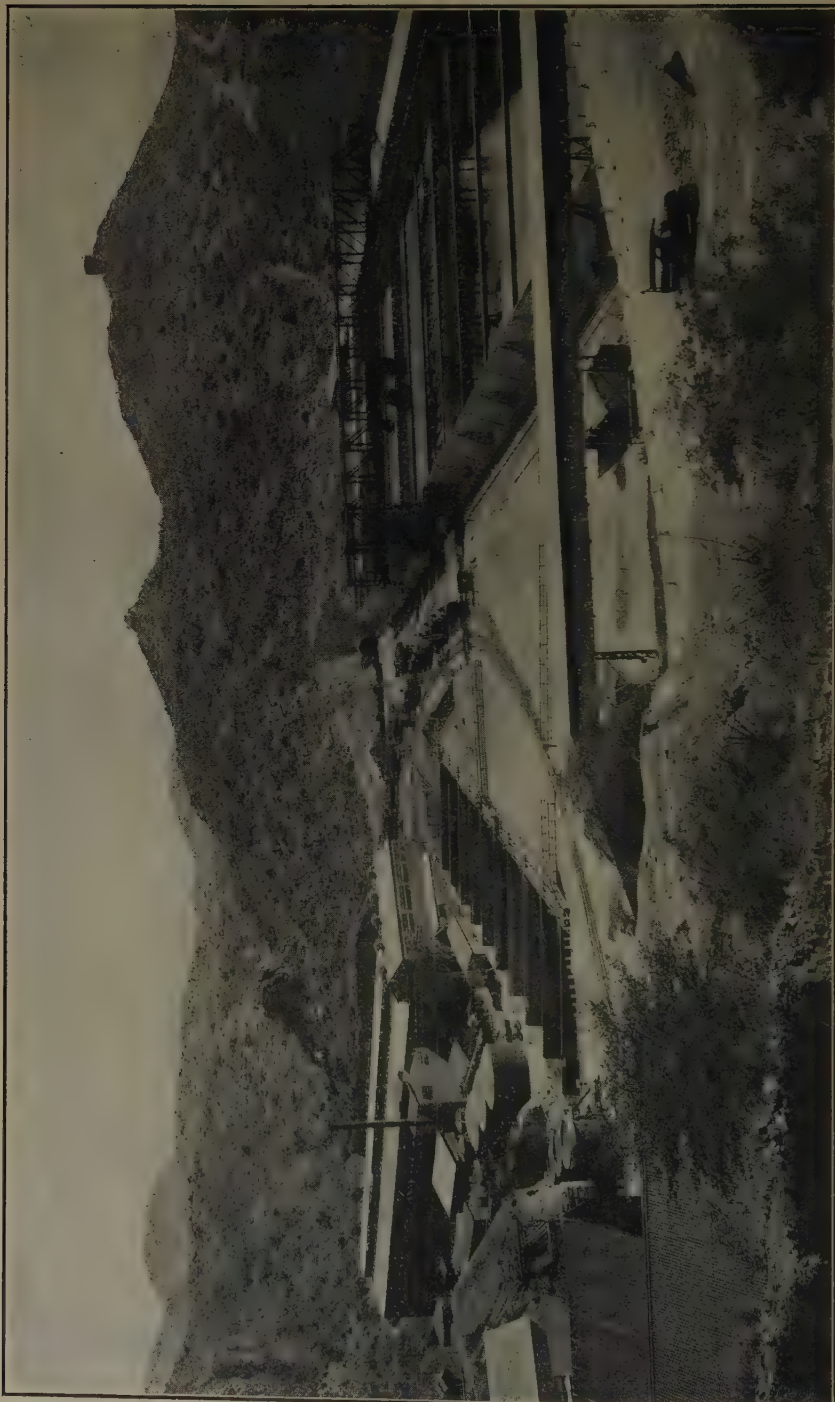

FIG. 4.—INSPIRATION LEACHING PLANT.

moves forward 2½ ft., until it reaches the edge of the tank, when the motion is reversed and the bridge travels toward the opposite edge. In this way the tank is filled with ore evenly over its entire area. The ore bed rises about 3 ft. for each pass of the bridge over the tank. By bedding in this manner, practically all rolling or segregation of the material in the charge is prevented. The screen analysis of the ore is given in Table 4.

TABLE 4.—*Screen Analysis of Ore Bedded*

Mesh	1930	1931	Mesh	1930	1931
+ 3	15.0	17.8	+ 35	3.4	3.1
+ 4	15.2	15.7	+ 48	2.6	2.4
+ 6	13.4	12.8	+ 65	2.3	2.0
+ 8	9.5	9.6	+100	2.2	1.8
+10	8.7	8.6	+150	1.4	1.4
+14	6.7	6.0	+200	1.1	1.0
+20	5.0	5.1	−200	9.0	8.9
+28	4.5	4.2			

After bedding, the ore is covered with a solution strong enough in acid to avoid neutrality at any point within the charge; otherwise a decided loss in iron content would occur through the precipitation of iron salts, and a high copper content of the tailing usually accompanies such precipitated iron. Between 175,000 and 200,000 gal. of solution is required to cover the ore. Percolation of solutions is upward. Each tank is provided with a vertical screw-type lead pump, which receives the overflow solution from the preceding tank and pumps it underneath the filter bottom and up through the ore. The solution overflows through a launder and goes to the suction of the pump serving the next tank, which permits the tanks under acid treatment to be in series. The circulation on each tank is approximately 1200 gal. per minute.

The cycle is 13 days; eight tanks being under acid treatment, three being washed, one being filled, and one being excavated. The strong acid solution from the tank house, in which sulfuric acid and ferric sulfate have been regenerated and to which the required amount of new acid has been added, is delivered to the oldest ore under acid contact. This solution travels from tank to tank, constantly being reduced in solvent strength and increased in copper content, until it emerges from the newest ore and flows thence to the tank house.

The principal difference in operation between the Inspiration plant and other large leaching plants is that at Inspiration a deliberate effort is made to maintain the ferric sulfate concentration at a point high enough to dissolve the large amounts of copper contained in the ore as chalcocite, although a high concentration of ferric sulfate is a decided detriment to

electrolytic precipitation efficiency. This also necessitates the carrying of a high percentage of total iron in solution.

Manipulation of solution and solution strengths to preserve an economic balance between the ferric sulfate necessary for sulfide extraction and avoidance of the penalty incurred by ferric iron in electrolytic precipitation efficiency presents a delicate point in operation. As the ore is changing constantly in grade, particularly the relation of sulfide copper to oxide copper, the economic limit of solvent strengths must also change. General practice has been to hold the average strength of ferric sulfate during the leaching of any one charge at 7.5 grams per liter, with the maximum strength at approximately 10.0 grams per liter, and the minimum strength at approximately 5.0 grams per liter. If the sulfide copper content of the ore is high, the reduction of ferric sulfate in leaching solutions is naturally more pronounced and a larger quantity of ferric sulfate is consumed. The necessary strengths, however, are maintained by increasing the volume of solution to the leaching division. Should the sulfide content be low, or the acid contact time be increased because of decreased production, the volume of solution pumped to the leaching division is reduced, with a consequent reduction in strength of ferric sulfate to the tank house. The consumption of ferric sulfate is approximately the theoretical amount required for dissolving the sulfide copper.

Prior to July of 1931, all concentrations of acid reported were given as total acidity. Since that time both free acid and total acidity have been given.

Acid Consumption

A major cost item is sulfuric acid. Before the slimes were removed, the acid concentration was always held higher than necessary for the extraction of soluble copper. Experimental work demonstrated the fact that even though the solution coming from the new ore had an acid reaction, it was necessary to have sufficient strength to avoid neutrality at any point within the charge of ore. After the slimes were removed, it was found that the better percolation avoided pockets and dead areas, which, in turn, permitted the use of a lower acid concentration, leading to a lower acid consumption. The acid consumption, from 1927 through 1931, was as follows:

	1927	1928	1929	1930	1931
Pounds 60° Bé acid per ton of charge........	31.4	26.5	33.3	23.4	23.3

There is a limit, however, to low acid concentrations, governed by operating conditions in the tank house, which will be referred to later.

From the standpoint of acid consumption, the ratio of production of electrolytic copper to production of cement copper plays an important

part. To maintain an iron balance in leaching solutions, a definite amount of cement copper must be made or iron must be added in some other manner. Acid is regenerated in the tank house with the precipitation of electrolytic copper, while acid is consumed in the precipitation of copper as cement copper. Generally, only low-grade wash waters are sent to the cementation division.

Washing

The washing of the ore, to avoid iron losses as well as copper losses, constitutes a very important phase in the operation. This may be pictured in the following manner (1931 results): A tank of 9000 dry tons when discharged as tailing will carry 10.3 per cent, or 927 tons, of moisture. As the final acid-solution drain averages 21.6 grams per liter of copper, a simple calculation shows that the amount of copper to be washed out would be 40,046 lb. per tank, or per day. This represents 19.45 per cent of the daily production.

Three days are used for washing, and every effort is made to wash out as much of this dissolved copper as possible. Ten washes in all are used. The first five are called regular washes and are systematically advanced, the first wash going to the main solution system, the second wash becoming the first wash on the next tank to be washed, and so on in the sequence noted. The fifth wash is taken from the cementation stock-solution tank and becomes the fourth wash on the next tank to be washed.

Four cementation-solution washes follow, and these washes likewise come from the cementation stock-solution tank. The used water is drained to a surge tank that supplies the cementation launders. Practically all of the copper is precipitated, the total iron content of the solution is increased, and acid and ferric sulfate are consumed. The cementation tailing solution goes to the cementation stock-solution tank for further use. Fresh water is used for the last, or tenth, wash.

Normally, a batch wash is used; that is, the entire wash water is pumped on to the ore, circulated and drained. At times, when a leaching tank has a short tonnage, the amount of wash water to fill the tank to its overflow would be so great that an unbalanced wash water volume would exist, therefore a piston or displacement wash is substituted. All conditions considered, batch washing is thought to be more positive if there is any doubt as to the rate of percolation through the ore.

The soluble iron in the ore amounts to less than ½ lb. per ton. The test-plant results over a long period showed that an average of only ⅓ lb. of iron per ton of ore was actually dissolved. As it is desired to carry about 20 grams per liter of total iron in the leaching solutions, the solution from the cementation system serves as an excellent means of controlling the total iron content. The amount of cement copper made has a direct bearing upon the iron put into solution by this method.

The only plant discard is the moisture in the tailing, therefore the wash-water volumes and wash-water advances are carefully watched. Every effort is made to add the fresh water as a final wash, as it is most valuable at this point. The volume balance for 1931 is shown in Table 5.

Table 5.—*Volume Balance*, 1931

	Gallons per Ton of Ore		Gallons per Ton of Ore
Entering moisture (in ore)......	16.49	Moisture in tailing..............	24.73
Fresh water added as wash......	15.87	Evaporation per ton per day.....	7.63
Total incoming water.........	32.36		32.36

When operating seven days per week, a normal wash-water volume would be between 125,000 and 150,000 gal. When operating at a reduced capacity, this volume can be increased, because the daily evaporation remains fairly constant.

Solution Heating

From October to May, it is necessary to heat the solution overflowing from the tank house on its way to the leaching system; otherwise a satisfactory extraction of the sulfide copper would not be obtained. This heating is accomplished by steam at 5-lb. pressure flowing through 1-in. chemical lead pipes having a $\frac{3}{16}$-in. wall, which are immersed in lead-lined tanks through which the solution flows. See Table 6.

Table 6.—*Heating Plant Data*, 1931

	To Tank House Deg. C.	To Heating Plant Deg. C.	To Leaching Tanks Deg. C.	Atmosphere Deg. C.
January......................	31.2	39.0	46.3	9.7
February.....................	34.4	40.7	47.1	12.3
March........................	33.4	39.4	45.8	15.1
April........................	33.4	39.1	44.5	18.9
May..........................	33.5	39.6	44.3	23.1
June.........................	33.2	40.8	40.8	27.4
July.........................	36.5	43.8	43.8	30.0
August.......................	36.6	43.0	43.0	27.2
September....................	35.0	41.1	41.1	25.9
October......................	33.9	40.5	42.0	20.1
November.....................	33.2	40.0	43.9	12.4
December.....................	34.6	40.9	45.5	9.4

In all, 24,738 ft. of pipe are used for this purpose. Two 500-hp. oil-fired boilers furnish the steam. One-half of the heating surface is used

on the inflow to the tank house and one-half of the surface is used on the outflow. Vacuum pumps are used to return the warm condensate to one of two boiler-feed tanks, where it is tested for acidity before entering the boiler. Whenever acid appears, the condensate is discarded and the leaky coil blocked off.

The raising of solution temperature may either precede or follow electrolytic precipitation. This gives a means of controlling the ferric sulfate balance, as will be mentioned under the description of the tank house. Operation of the experimental plant demonstrated the fact that a cold solution, approximately 20° to 22° C., would give a sulfide copper extraction, with a normal ore and under normal conditions, of between 40 and 50 per cent. By raising the maximum temperature on the oldest ore to 42° C., the minimum temperature would rise to 28°, giving an average temperature on any individual charge of approximately 35° C. This condition, other things being equal, produced a 75 per cent average sulfide copper extraction.

During the four summer months, when the heating plant is closed, the average solution temperature is about 38° C. The problem of heating to a higher temperature in winter presents the question of an economic balance. Radiation losses above 40° in winter are large, and as a result the cost of raising the temperature one degree increases rapidly. Solution leaving the heating plant at 50° C. and pumped through an insulated 12-in. lead pipe line 1500 ft. long will give an average temperature to the coldest ore of 42° C.

Tailing Disposal

After a tank of ore has been leached, washed and drained, it is ready for excavation. A Wellman-Seaver-Morgan bridge-type excavator spans the leaching tanks with the bucket carriage traveling across the long dimension of the tank and the bridge moving the entire length of the 13 leaching tanks. The operator rides on the bucket carriage. The bucket is capable of lifting approximately 17 tons of wet tailing. It travels to the end of the excavator and discharges its load through a hopper into air-dump, standard-gage railroad cars. Three buckets usually constitute a carload, and the number of cars to a train will vary from six to ten, depending upon dump conditions.

Two 8-hr. shifts are required to excavate a tank. Two tailing trains operate on the first shift, when the digging is easy and fast, but only one train is used on the second shift. During the first year of operation, owing to the inaccuracy of control of the hoist motors, the bucket lips tore up the filter bottoms so badly that 4 by 6 in. timbers, laid on the 4-in. side, are now placed over the filter boards, running parallel to the long dimension of the tank on 3-ft. centers, to receive the wear of this operation. This necessitates the leaving of a 6-in. layer of the tailing in the

bottom of the tank at all times. These timbers must be replaced at intervals of from five to six months.

The tailing dumps are from one to three miles from the leaching plant. They are referred to numerically. The No. 3, or main dump, is nearly 300 ft. high. As the height of this increased, the tailings did not carry enough moisture to permit them to slide to the toe of the dump, and a bulge would develop on the face of the dump, which would slide when least expected, taking with it the track and any equipment that happened to be on it (Fig. 5). The addition of a small quantity of water through a 1-in. hose prevented these bulges from forming and completely remedied

Fig. 5.—Slide on tailings dump at Inspiration leaching plant before present practice of washing down with water.

this condition, causing the slope to be uniform from the top to the toe of the dump. As the dump is built out, the track is moved out by a mechanical track shifter.

Slime-treatment Division

The removal of fines from the leaching-plant feed had received considerable attention for some time before the large-scale experiments of 1928. Numerous data, obtained in 1919, were available on this subject. Wet classification had been put off to avoid (1) the moisture admitted to the leaching process with the washed sands, and (2) the mechanical difficulties that might arise in handling the wet sands in conjunction with the dry material.

To date, the solution volume balance within the leaching plant has been maintained, though the margin is small. The handling of the wet

sands has also been solved as follows: The dry discharge from a cross conveyor is first placed on the main belt going to the leaching tanks. This load is leveled on the belt by means of a small V-scraper. The wet sands are discharged on to the dry material and another cross conveyor discharges dry material on top of the wet. At the first conveyor junction, the entire load is turned over and mixed. The moisture coming in with the classifier sands also serves to settle the dust.

Following the decision to remove the slime from the leaching-plant feed, tests on agitation leaching and settling were made by the leaching department, and flotation tests by the concentrating department, to determine a possible treatment plant for this material.

Governed by the fact that time would not permit the design of new or untried equipment, and also that the concentrator division had ample unused equipment, only standard practice was given serious thought, and the following type of plant was constructed and placed in operation.

Flotation

The pulp, with the addition of standard flotation reagents, enters an Inspiration-type flotation cell direct, as no grinding is necessary. The resultant concentrate is settled in Dorr thickeners before filtration, while the tailing goes to Dorr thickeners, formerly used by the concentrator, which precede the slime-leaching plant. Results are shown in Table 7.

TABLE 7.—*Results of Flotation Treatment from Yearly Reports*

	1930	1931
Dry tons treated	154,909	146,468
Dry tons per day	545.5	569.9
Solids in feed, per cent	17.71	20.35
Minus 200-mesh in feed	87.5	85.8
Feed		
Oxide copper, per cent	1.285	1.156
Sulfide copper, per cent	0.346	0.389
Total copper, per cent	1.631	1.545
General concentrates, percentage of copper	45.341	44.300
Tailing		
Oxide copper, per cent	1.187	1.072
Sulfide copper, per cent	0.138	0.169
Total copper, per cent	1.325	1.241
Recovery		
Oxide copper, per cent	7.63	7.94
Sulfide copper, per cent	60.12	56.78
Total copper, per cent	18.76	20.21
Copper per ton of feed, lb	6.12	6.24

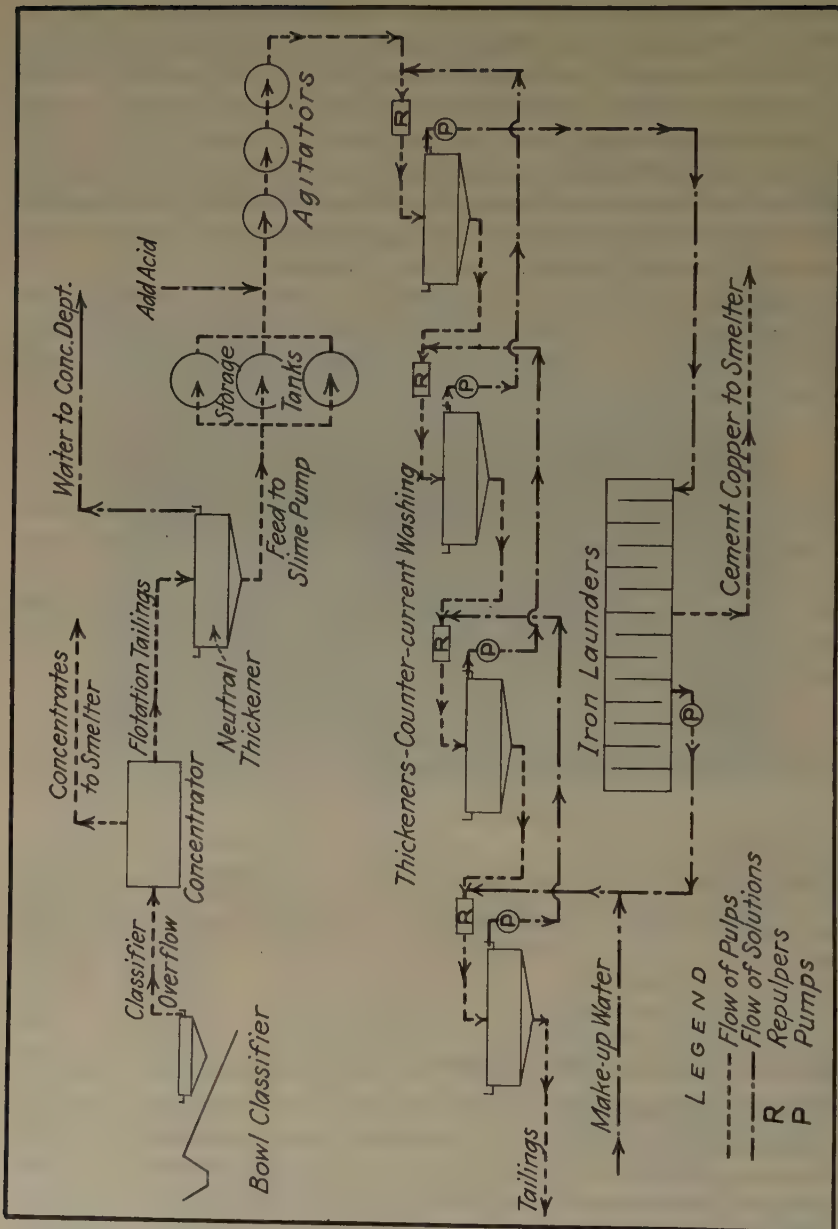

FIG. 6.—FLOW SHEET OF SLIME TREATMENT PLANT, INSPIRATION CONSOLIDATED COPPER CO.

Slime-leaching Division

Fig. 6 shows the flow sheet of the slime-treatment plant. Three Dorr slurry mixers, each 36 ft. in diameter and 20 ft. deep, are used for storage of the thickened slime. The capacity of these mixers is 750 tons of dry slimes in a pulp carrying 33⅓ per cent solids. The pulp entering the storage tanks is sampled for assay and the percentage of solids determined. This pulp sample represents the feed to the slime-leaching plant. From the storage tanks, the pulp is drawn continuously to the Dorr agitators. Acid is added with the pulp entering the first of three agitators after being accurately measured in unit tanks which are filled from the main acid storage.

The agitators consist of circular, flat-bottomed, wooden tanks, lead-lined, with a one-inch wooden lining, loosely fitted, to protect the lead. The agitating mechanism is carried by a superstructure resting on the tank walls. The mechanism is made up of a central hollow steel shaft above the pulp level, and wood, treated with creosote, below. This wooden structure is built so as to serve as an air-lift, the air being supplied through the hollow shafting to lead pipes.

Two raking arms are attached to the lower end and two troughlike distributing arms at the upper end. The wooden structure is fastened with bolts completely incased in lead, and the blades are of steel, covered with rubber.

TABLE 8.—*Metallurgical Results at the Slime Plant*

	1930	1931
Feed		
Dry tons	104,037	145,001
Operating days	220	290
Tons per day	472.9	500.0
Solids, per cent	38.1	38.5
Minus 200-mesh material, per cent	86.9	86.0
Oxide copper, per cent	1.175	1.068
Sulfide copper, per cent	0.161	0.175
Total copper, per cent	1.336	1.243
Tailing		
Oxide copper, acid-soluble, per cent	0.040	0.061
Oxide copper, water-soluble, per cent	0.022	0.035
Sulfide copper, per cent	0.101	0.122
Total copper, per cent	0.163	0.218
Extractions		
Oxide copper, per cent	94.72	91.01
Sulfide copper, per cent	37.27	30.29
Total copper, per cent	87.80	82.46
Copper per ton, lb	23.46	20.50
Pounds 60° Bé acid per ton ore	108.3	100.3

FIG. 7.—AIRPLANE VIEW OF SLIME-LEACHING PLANT, INSPIRATION CONSOLIDATED COPPER CO.

The agitators are 20 ft. in diameter and 14 ft. deep, with a combined capacity of 9918 cu. ft. A speed of 6 r.p.m. of the agitator arms is used. As the agitators operate in series, the pulp with acid added enters the first agitator. The acid contact, or agitation time, ranges from 5 to 6 hr., depending upon the tonnage to be treated. The recovery of the dissolved copper leached from the slimes by agitation is accomplished by counter-current washing. Metallurgical results at the slime plant are shown in Table 8 and an airplane view of the plant is given in Fig. 7.

Thickeners

Four Dorr traction thickeners 150 ft. in diameter are used. The tanks are of reinforced concrete with gunite surface. The traction arms are of treated wood below the solution, fastened with bolts completely incased in lead, and the rakes are of rubber-covered steel. Each thickener has two triplex Dorrco diaphragm suction pumps, rubber-lined so as to be acid-resisting. Repulpers, or mixers, precede all thickeners or wherever thickened pulp is mixed with clear overflow solution. Lead lines are used for conveying the various solutions, while wooden launders, painted with an asphaltic paint, are used for transportation of pulps. The recovery of copper dissolved was 98.16 per cent in 1930 and 96.7 per cent in 1931. Table 9 gives the report for 1931.

TABLE 9.—*Slime-plant Metallurgical Report*, 1931

Overflow	Solution Analysis, Grams per Liter			Thickener Discharge, Per Cent Solids
	Copper	Acid	Total Fe	
1st thickener	2.73	4.7	3.5	46.8
2d thickener	1.19	2.7	5.4	44.5
3d thickener	0.53	2.3	6.4	43.7
4th thickener	0.21	2.1	7.0	43.7
From cementation launder	0.00	2.1	7.8	

Cementation Launders

The launders, in principle, are a duplication of those at the leaching plant. At the leaching plant, however, the launders are in one group, with the iron storage at one end and the trommel for washing at the other end. At the slime plant, the launders are divided into two groups with the trommel in the middle and the iron storage at both ends. This greatly shortens the crane travel and speeds up operation. Seven double-section launders, 40 ft. long and 20 ft. wide, with an average solution depth of 5 ft., are used in this service. Baled tin cans are used as a precipitant. See Fig. 8.

A traveling gantry crane spans thee ntire plant. By means of an electromagnet, the cans are unloaded from railroad cars and placed in storage or put directly into the iron launder. When a section of the launder reaches a point where precipitation has practically stopped, the solution is drained through weep holes and excavation with a clamshell bucket is started.

FIG. 8.—IRON LAUNDERS AT THE INSPIRATION SLIME PLANT.

In separating the undissolved cans from the cement copper, a trommel 12 ft. long and 5 ft. in diameter, revolving at approximately 13 r.p.m., is used. Holes ¾ in. in diameter permit the cement copper to fall into a sump, while the undissolved cans come out of the lower end of the trommel to another sump and are returned to the launders. Water entering through a pipe extending into the trommel washes the cement copper

TABLE 10.—*Combined Results from Leaching and Slime Plant*

	1930	1931
Original feed to leaching plant		
Oxide copper, per cent	0.704	0.655
Sulfide copper, per cent	0.536	0.665
Total copper, per cent	1.240	1.320
Combined tailing from leaching plant and slime plant		
Oxide copper, per cent	0.022	0.025
Sulfide copper, per cent	0.093	0.140
Total copper, per cent	0.115	0.165
Combined extraction		
Oxide copper, per cent	96.875	96.183
Sulfide copper, per cent	82.649	78.947
Total copper, per cent	90.726	87.500
Copper recovered per ton, lb	22.50	23.10

through the holes into the cement-copper sump. The water overflows to a pump and returns to the trommel. The cement copper is loaded from the sump with the clamsheel bucket to cars, and is shipped to the smelter.

Combined Extraction Results from Leaching Plant and Slime Plant

Although the leaching and slime plant results are reported separately, the combined results actually represent the copper extracted from the ore. Because of the difference in treatment, and inventory material under treatment, it is difficult to match the individual charges or lots correctly. However, the period results are approximately correct, as shown in Table 10.

Tank House

The tank-house personnel consists of 50 men, which includes all help for the starting-sheet section as well as for the commercial section. The tank house is capable of producing 200,000 lb. of copper per day.

General Electrical Data

The generating equipment has a capacity of 36,000 amp. at 320 volts. It consists of three units, each having two generators, centrally driven by a 5450-hp., 500-r.p.m., synchronous motor.

As the tanks are arranged in two parallel lines, great precaution is taken to avoid tank short circuits, as these result in a direct loss of power. An 11-ft. length of 4-in. rubber hose is inserted in the solution feed line to each tank, and a 16-ft. length of 12-in. rubber hose in each pump suction avoids short circuiting at this point. Each tank overflow drops through a 6-in. tile pipe to a central distributing launder.

The tank house consists of 120 commercial tanks and 20 starting-sheet tanks.

Starting-sheet Division

Considerable experimental work has been carried on in an endeavor to make a satisfactory starting sheet in the commercial division. To date, the conclusion has been reached that it is possible to make starting sheets from commercial solution, but the operation would be very uncertain and it is not recommended. As the tank, house metallurgy must be regulated to suit leaching conditions, there are times when the circulating solution is not suited for the making of a good starting sheet.

This division is, in reality, a small refinery of 20 tanks; each tank has a capacity of 95 blanks (cathodes) and 96 anodes, making a total of 1900 blanks, capable of making 3800 starting sheets per day. The loops must

be deducted from this production, so 15 per cent is usually allowed for loops, blanks not stripped, and scrap. This gives a capacity of 3230 finished starting sheets per day. Production, however, is regulated to meet the requirements of the commercial department, which in turn is dependent upon the desired production of copper. The blanks are of refinery type, of rolled copper, with a groove on each side and on the bottom to produce a clean-cut sheet.

Electrodes are spaced 4½ in. from center of cathode to center of cathode, and blanks are stripped every 24 hr. A current density ranging from 14 to 16 amp. per square foot is used, and the average starting sheets weigh from 11 to 12 lb. each before looping. The solution circulation rate is 15 gal. per tank per minute, and the solution carries from 30 to 40 grams per liter of copper and 120 to 150 grams per liter of acid. Accumulation of impurities is prevented by bleeding refinery electrolyte to the commercial division. Two hot wells maintain a temperature of approximately 55° C. From 10 to 30 lb. of glue are added per day, at the discretion of the operator.

Refinery mud, carrying gold, silver and copper, is washed to settling tanks, dried in open pans and shipped.

The finished starting sheet is 44 in. long by 42 in. wide. The loops, which are cut with power shears, are 20 in. long by 4 in. wide before looping. On the ends of the starting sheets 2 in. are allowed for punching. This gives a finished sheet that allows 42 in. for submergence, with the entire loop and punching above the solution level.

A new, rolled-copper blank is thoroughly cleaned before use, to remove any oxide, and amalgamated with mercuric nitrate. It is then oiled more heavily than normal and placed in operation. The first sheets to be stripped are usually not very good, but after a few days in operation, there is no more trouble. If a blank is to be removed and stored, it is not stripped. In this way the surface of the blank is protected from oxidation.

Blanks are removed in groups of 14; and the hooks on the lifting basket are spaced so as to lift alternate blanks, which permits more working room for stripping. At first, a steel tool made of small-size, hexagonal steel, properly sharpened and curved, was used for starting and stripping the starting sheets from the blanks. After 18 months of operation, a wooden tool made from an ordinary pick handle, with a steel end suitably attached, to be used as a starter, was substituted, as this did not scratch the blank. A smooth blank naturally a'lows easier stripping and yields a higher percentage of satisfactory starting sheets, with less scrap.

After stripping, the blank is touched with amalgam solution wherever the steel starting tool may have scratched it. A cheap grade of oil is applied with cloth swabs, and the blank is then returned to the refining tanks.

Blister-copper anodes, cast from converter copper, are received from the International smelter. These range in weight from 900 to 1100 lb. each. The set side of these castings usually is rough with occasional slag spots, but this type of anode has been handled with comparatively little trouble, although new anodes do not consistently produce a good sheet for the first few days. After a tank begins to produce good starting sheets, however, it will often produce 100 per cent for days at a time.

The plant is not equipped to work up odd lots of scrap anodes of varying sizes, so when a tank shows some scrap the entire tank is scrapped. To date, 24.3 per cent of anode scrap has been made. This material is shipped back to the International smelter for remelting and casting.

FIG. 9.—TANK HOUSE, INSPIRATION LEACHING PLANT.

Commercial Division

The commercial division consists of 120 electrolytic tanks (Fig. 9), 33 ft. long, 4 ft. 3 in. deep, and 4 ft. wide. These are divided into 8 banks of 15 tanks each, the electrolyte flowing in series through the banks. Each bank has a solution circulation of approximately 1200 g.p.m., or 80 gal. per tank per minute.

Each tank has 95 cathodes and 96 anodes, spaced 4 in. from center of cathode to center of cathode and placed perpendicular to the direction of solution flow. This makes a total of 11,400 commercial cathodes and 11,520 lead anodes.

Originally, a 4½-in. spacing of electrodes was used. It was found that a current density of 11 to 12.5 amp. per square foot produced better results than higher current densities, and with less attention. A 4-in. spacing, which is now being used, permits greater cathode area and will give a greater unit production at a lower current density. When operating with the current density mentioned, this spacing is the more

economical. It was felt that the greater area would give a larger precipitating capacity, but inspection difficulties have greatly increased with this closer spacing and it is doubtful whether this advantage exists.

The original anodes were made of ¼-in., 4 per cent antimony lead, 38 by 40 in. submerged, and supported by copper bars. Each anode was punched for eight porcelain insulators, which tend to prevent short circuiting, but now only five of these are used. These ¼-in. anodes warped badly and they were first straightened by batting, with the result that the lead flowed badly. A multiple straightening press, operated by air pressure, was next installed, but the anodes were too light and it was decided to change to a heavier type. At present, all commercial tanks are equipped with ½-in. lead anodes containing 8 per cent antimony. These were cast and fabricated at this plant and are in excellent condition after more than two years of use.

TABLE 11.—*Tank-house Results*

	1927	1928	1929	1930	1931
Current density	14.2	11.8	14.6	13.3	11.9
Kilowatt-hours a. c. per pounds copper precipitated	1.641	1.540	1.695	1.615	1.431
Cathode efficiency, per cent	61.4	64.7	61.8	64.1	67.9
Anode efficiency, per cent	41.1	43.3	60.7	52.0	54.0
Copper, grams per liter, in	28.2	23.4	28.2	33.1	26.4
Copper, grams per liter, out	22.3	18.2	23.1	26.6	21.6
Total acidity, grams per liter, in	41.4	30.6	25.5	26.9	33.8
Total acidity, grams per liter, out	56.6	45.0	42.4	38.4	41.1
Ferric iron, grams per liter, in	5.9	5.3	4.5	3.1	4.0
Ferric iron, grams per liter, out	10.2	9.3	10.0	9.1	8.6
Total iron, grams per liter	17.3	16.0	18.6	16.4	20.0

The starting sheets are threaded (insertion of copper supporting bar), painted and straightened by contract labor. The painting is done just at the solution line with a single stroke of a brush, using a cheap, semi-acid-resisting paint. A sheet not painted in this manner will cut off at the solution line in about five days. The sheets are then grouped on racks, moved by the crane opposite the tank where they are to be used, and are placed in the tank by hand.

Each cathode is straightened at the end of 24 hr. Additional straightening or the removal of sprouts is dependent upon operating conditions. After each movement, the cathodes are checked by meter men for poor contacts or short circuits. Both a voltmeter and a locally designed millivoltmeter are used to detect short circuits and faulty contacts. The latter consists of two contact points, accurately spaced in a small wooden frame that is placed on the cathode supporting bar, and the current flow is recorded in millivolts. This is a very sensitive means of detecting trouble.

Best results are obtained by pulling a finished cathode weighing between 90 and 100 lb. total weight. The cathode life, therefore, varies with the current flow. A five-day cathode is pulled for a maximum production. In pulling copper, one-fourth of a tankful is lifted at one time, conveyed to the wash tank and submerged in boiling water. From here the cathodes are dropped on to a steel rack with an inclined back, by moving the crane slowly backward and permitting them to knock themselves off the crane basket hooks. Two pulls are placed on each rack. A storage-battery truck with a low platform slides under the loaded rack, raises it and transports the load, first over the scales and then into railroad box cars for shipment. In the cars the cathodes are lifted by hand and stacked as desired.

The weight of copper from each electrolytic tank is recorded, and the efficiency figures for each day's production are arrived at by using ampere and voltage readings taken each shift during the life of the particular cathodes pulled. Starting sheets going to each tank are weighed, this weight being deducted from the total cathode weight to obtain the weight of copper precipitated. Tank voltage readings are taken each shift and carefully compared to the total switchboard voltage. In the generator room hourly ampere readings are recorded, and the kilowatt input and output of the generator station is measured by recording meters. All final figures from this department are based on copper production and power actually paid for. The latter figure includes line loss from substation to generators, and generator conversion efficiency, as well as all transmission, contact and minor losses.

The tank-house production, naturally, is dependent upon the amount of copper dissolved from the ore, and the current density is varied to satisfy this condition. Within certain limits, a reduction in current density is accompanied by a lower power consumption. For instance, within the normal working range of the tank house, all conditions being equal, the voltage decreases 0.1069 with a decrease of one ampere per square foot in current density. The power represents approximately 80 per cent of the total tank-house cost. Results of operation in the tank house are given in Table 11.

If the copper strength in the electrolyte is held too high, the copper in solution entrained in the ore to be washed will be high, resulting in the manufacture of excess cement copper, thus increasing the cost of production. Within certain limits the grade of copper in the electrolyte has little effect on precipitation results.

For the first four years of operation, the acid strength to satisfy the possible solution neutralization on the ore, with the subsequent precipitation of iron salts, was held sufficiently high so that no trouble was experienced from this source in the tank house. In the winter of 1930–1931, it was decided to try a much lower acid concentration for

leaching. An equally high oxide extraction was made, but this low acid strength to the tank house caused the precipitation of a basic salt on the anode which reduced anode efficiency materially. Although the tank house has operated many days with a free acid strength of 10 grams per liter in the entering solution with no poor results, for the sake of safety the low limit of free sulfuric acid to the tank house has been placed at 15 grams per liter. The acid strength must, therefore, be regulated to satisfy the acid consumption and oxide extraction in leaching and, at the same time, not less than 15 grams per liter going to tank house.

This entire process is built around the efficiency of oxidation of ferrous sulfate to ferric sulfate in the tank house. Ferric sulfate is extremely detrimental to tank-house efficiency. A variation of total iron in solution, together with the arrangement to heat solution, either preceding or following electrolysis, constitutes a partial control of the ferric sulfate balance. Further, an anode freshly cleaned will produce the highest anode efficiency. The present anodes were placed in operation in March, 1929, and in January, 1931, after two years of continuous operation, the anode efficiency dropped from an average of over 60 per cent to about 35 per cent. Anode efficiency may be defined as the relation between the actual amount of ferrous sulfate oxidized to ferric sulfate per pound of copper precipitated to the theoretical amount. The only apparent change in operation was the lowering of the acid concentration. It was noticed that in addition to a very hard scale, which had been forming for some time and which tended to blind the anode, there was a small amount of hard, yellowish precipitate on the anode.

Anode cleaning was new and some little time was lost in arriving at a satisfactory method for cleaning so many electrodes. Experimental work showed that by reversing the current on an anode, that is, using the anode as a cathode, in a water-acid bath, the anode could be cleaned easily. Several standard tanks were set aside for this purpose. After 60 minutes' treatment (although a longer time made the cleaning easier and better), in a water-acid bath, the acid strength running as high as 300 grams per liter, and the current density from 25 to 30, the anodes were easily cleaned by means of a brush and jet of water. By cleaning the anodes every two weeks, it is possible to maintain an average anode efficiency of approximately 75.0 per cent. Average figures show that a voltage drop of 0.05 volts takes place after each cleaning.

It was concluded that the yellowish precipitate could be avoided by maintaining the proper acid concentration. The scale, however, will continue to form with operation and will have to be cleaned off at intervals, the frequency of which depends upon the desired strength of ferric sulfate or, in other words, anode efficiency.

Sampling of the cathode copper is done once a month, which serves as a control and a check on refinery results. A typical analysis of electrolytic

copper of March, 1931, is as follows: Cu, 99.87 per cent; Fe, 0.021; Ni-Co, 0.0006; Zn, tr; Sb, 0.0005; As, tr; S, 0.045; Se and Te, tr; Cl, tr.

Electrical Losses

Because of the single-circuit tank house, the circuit is opened at specified intervals, at some point distant to the generators, and the voltage brought to an operating level in order to determine losses. The leakage is measured directly in power, and it has never exceeded 1.0 per cent with a voltage as high as 350 volts, which is a 9.4 per cent generator overload. A complete survey of the tank house shows an electrical loss of 0.62 per cent through the copper buses, and a contact loss of 0.87 per cent.

Cementation Launders

Nine double-section cementation launders, 60 ft. long and 20 ft. wide, with an average solution depth of 5 ft., are used in this service. Baled tin cans are used as a precipitant. Only copper from wash waters is precipitated as cement copper, and as the surge tanks are of sufficient size, both preceding and following this plant, the operation is continuous. The solution flow in the tanks is in series and the distance of travel is regulated so as to remove practically all of the copper and, at the same time, lose as little of the acid as possible.

As at the slime plant, a traveling gantry crane spans the entire plant, and, in general, the entire operation is the same.

The total amount of cement copper produced is practically the same as the amount of blister copper used for starting sheets, and, therefore, approximately amounts to an exchange of copper with the smelter.

There being no discard at this plant, the purpose of iron precipitation is to assist in maintaining the desired iron strength in leaching solutions, and allowing a cleaner wash of leached tailing with a minimum addition of fresh water. Table 12 gives some general results. The consumption of iron seems very high, but this is caused by high percentages of acid and ferric sulfate in the solution going to the launders. This division has produced as high as 1,000,000 lb. of copper as cement per month with one craneman and three Mexican laborers per shift, working two shifts per day.

Table 12.—*Iron Precipitation*

	Copper Precipitated per Ton Ore Leached, Lb.	Iron Consumed per Pound of Copper, Lb.
1927	3.16	1.924
1928	2.79	2.054
1929	4.34	2.048
1930	2.77	1.868
1931	2.33	2.209

The Metallurgical Plant of the Andes Copper Mining Company at Potrerillos, Chile

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The property of the Andes Copper Mining Co. is in the central part of the Republic of Chile, Province of Atacama, about 75 miles to the eastward from the port of Chañaral. At this point the Andean system is made up of two principal ranges, and the mine and metallurgical plant are both on the westerly slope of the outer or western range. Mean elevation of the mine is 10,500 ft., while at the plant the elevation ranges from 9445 to 9700 ft. A meter gage railroad extends from the company's port at Barquito, near Chañaral, to the reduction plant at Potrerillos. This road has a total length of 94 miles, of which 57.79 miles are owned by the Andes Copper Mining Co. and the remainder of the road is controlled by the State.

Development of the Potrerillos mine and installation of the metallurgical equipment presented unusual difficulties due to the arid nature of the country, high altitude and remoteness of the location from any considerable center of population. As a primary step it was necessary that railroad transportation be provided in order that mine development and plant construction could go forward on an economical basis.

Provision of an adequate supply of water for domestic and metallurgical uses involved the construction of approximately 178 miles of pipe lines, all of which have their sources at altitudes ranging from 12,000 to 14,000 ft. above sea level and traverse rough and difficult terrane. Water for metallurgical use is brought to Potrerillos from the Ola River, a stream originating in the high cordillera to the eastward from the property. It is delivered by means of a steel pipe line, 32 miles long, having diameters ranging from 22 to 36 in. Domestic water is supplied to the various camps through a system of pipe lines aggregating 146 miles in length, the Ola water being unfit for this purpose. These conductors gather water from numerous springs in the high mountains to the eastward from Potrerillos and deliver it to the mine, the reduction works and the port at Barquito.

Extensive permanent housing facilities had to be provided as it was necessary to care for more than 3000 workmen and employees at the

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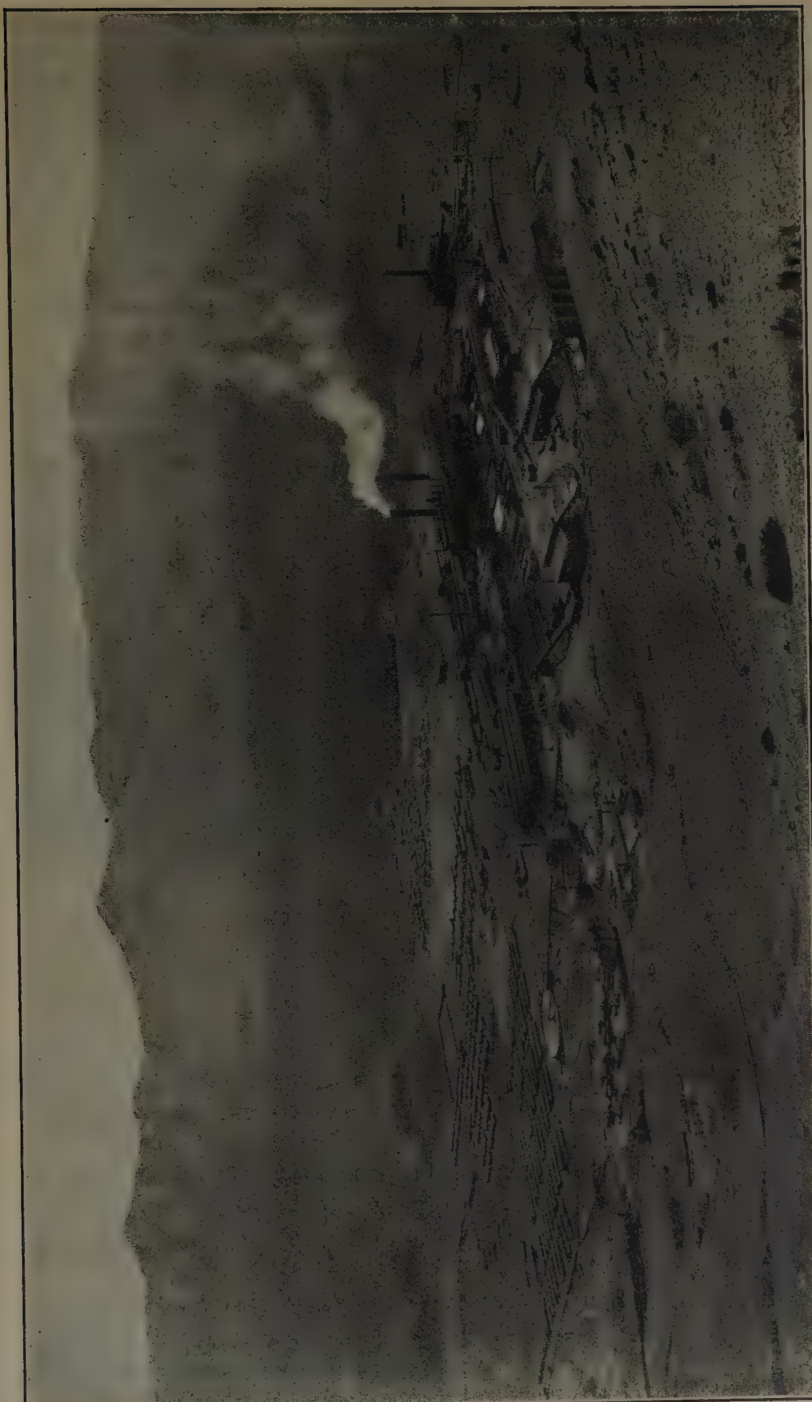


FIG. 1.—PANORAMIC VIEW OF THE METALLURGICAL PLANT AND TOWN SITE, ANDES COPPER MINING COMPANY.

various camps, including the mine, Potrerillos, Llanta (a division point on the Potrerillos Railway) and at Barquito.

Power needed for general use is generated at three different points; namely, the Barquito steam-power station, the Montandon hydroelectric installation, and the waste-heat steam plant at Potrerillos. In the Barquito station there are installed five B. & W. 1000-hp. boilers and four 7500-kw. steam-turbine driven generators. Current is delivered to Potrerillos at 88,000 volts over a double three-wire transmission line carried on steel towers. Total length of this line is 79.23 miles. The Montandon hydroelectric station is on the Ola pipe line and makes use of the part of the effective water pressure that is not needed for delivering

FIG. 2.—SMELTER AND ACID-PLANT BUILDINGS, ANDES COPPER MINING COMPANY.

the water to the reduction works. This station has a generating capacity of 1800 kw. Power is delivered to Potrerillos over a three-wire single-circuit transmission line 9.57 miles long, at 13,000 volts.

In the smelting plant waste heat is utilized for generation of steam and a 600-hp. Heine boiler is also available when needed. Two 3000-kw. generators are provided for utilizing waste-heat steam, and that from the auxiliary boiler.

The service department includes a machine shop, boiler shop and foundry, equipped with adequate machine tools for handling all necessary repair work. A carpenter and pattern shop, equipped with a full layout of wood-working machines, is also provided.

In the company's orebodies the copper occurs in both sulfide and oxide form. However, the cutoff between oxide and sulfide ore is sharp and there is little or no mixed ore. Because of the necessity for treating these two classes of ore, two separate metallurgical plants were con-

structed. They are interdependent only to the extent that the sulfuric acid for leaching the oxide ore must be produced through the roasting of sulfide concentrates, there being no other available source of cheap sulfur.

The reduction plant is on an alluvial shelf about 2½ square miles in extent, the surface sloping to the westward on a mean gradient of 10 per cent. The various metallurgical units are located so as to provide for a gravity flow of pulps and solutions generally throughout the plant. A number of gulches to the westward from the plant site provide ample area for impounding tailings.

Construction of the sulfide division was started early in the year 1925. The concentrator began work in the month of December, 1926, and the smelter in January, 1927. Construction of the oxide section was proceeding before the sulfide units were finished, and this division was ready for production in the month of May, 1928.

The mine and reduction works are connected by a meter gage railroad 6.215 miles long, which extends from the plant receiving bins to the underground storage bins at the mine. This road has a compensated grade of 0.3 per cent. Tunnels make up 53.50 per cent of its total length. Trains are made up of 28 cars each and are hauled by two 45-ton General Electric direct-current locomotives operating in tandem, which take their power from a third rail at 500 to 650 volts.

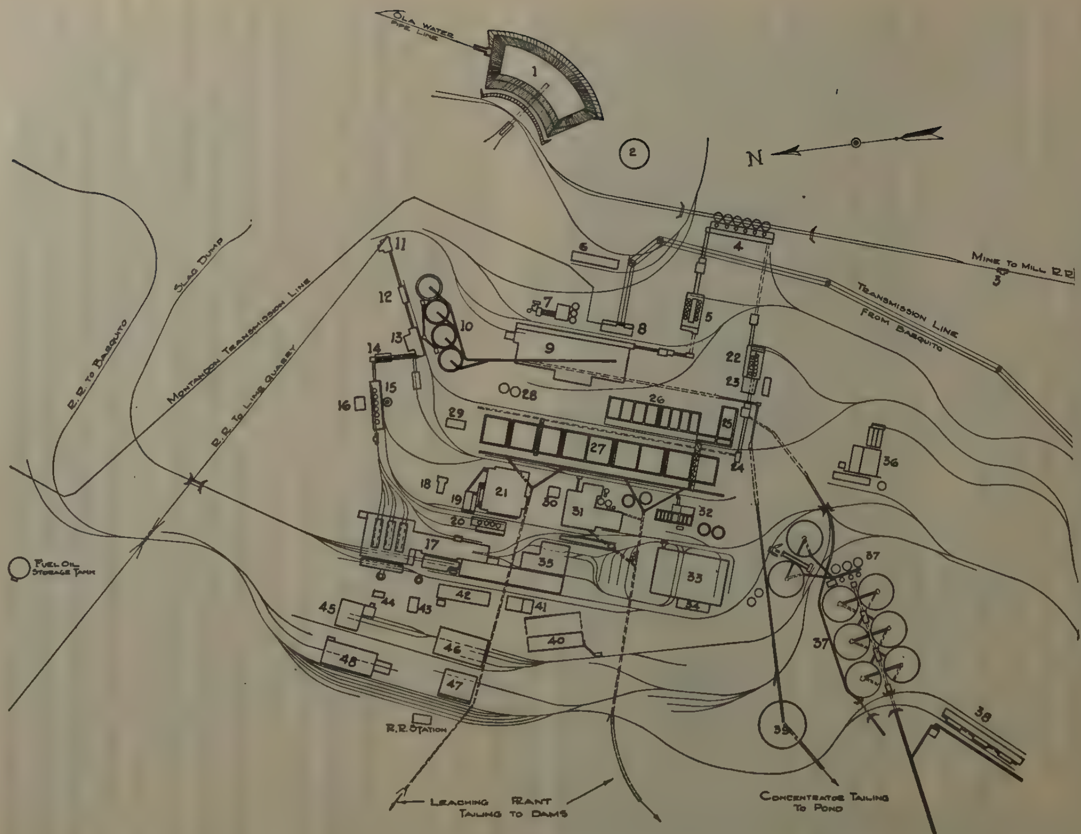
The ore is delivered direct to the receiving bins at the coarse-crushing plant, and a description of this unit, as well as other departmental installations at the Potrerillos reduction works, will be found in the following pages. A general plan showing the relationship of the various metallurgical units, railroad trackage, etc., is also included (Fig. 3) together with diagrammatic flow sheets pertaining to each of the principal divisions.

Sulfide Metallurgical Plant

The sulfide metallurgical plant includes a roaster plant, reverberatory furnaces and converters. The material treated consists of concentrates, oxide-plant secondaries and lime rock.

The concentrates are delivered from the Oliver filters of the concentrating plant to a 14-in. conveyor belt, which, in series with two other 14-in. belts, delivers the concentrates either to the roaster plant or to an emergency storage bin under the inclined conveyor between the filter and roaster plants.

Because it is necessary to produce two grades of concentrates, one having high sulfur and low arsenic content for acid making and the other having the remainder of the arsenic and copper in the form of a high-grade concentrate for direct smelting after drying, all the conveyor



1. Ola water reservoir.
2. Fresh-water reservoir.
3. Scale house.
4. Coarse-crushing plants.
5. Sulfide fine-crushing plant.
6. Flotation-reagent warehouse.
7. Lime-burning and hydrating plants.
8. Main electrical substation.
9. Concentrator.
10. Concentrate thickening tanks.
11. Limerock crushing plant.
12. Flux bin.
13. Concentrate filter plant.
14. Concentrate storage bin.
15. Sulfide roaster plant.
16. Cottrell substation.
17. Reverberatory and converter plants.
18. Mixed-acid plant.
19. Acid-plant Cottrells.
20. Acid-plant roaster.
21. Acid-plant generator units.
22. Oxide fine-crushing plant.
23. Oxide-fines washing plant.
24. Oxide sample tower.
25. Leaching-plant pump house.
26. Leaching-plant solution sumps.
27. Leaching vats.
28. Acid-storage tanks.
29. Laboratory sample mill.
30. Badenhausen boilers.
31. Purification plant.
32. Dechloridizing plant.
33. Electrolytic tank house.
34. Tank-house substation.
35. Fire-refining and casting plant.
36. Brick plant.
37. Slimes-leaching plant.
38. Cementation launders.
39. 200-ft. thickener for concentrator tailing.
40. Carpenter shop.
41. Lead shop.
42. Smelter power house.
43. Heine boiler.
44. Chemical laboratory.
45. Foundry.
46. Machine and electric shops.
47. General warehouse.
48. Boiler shop.

FIG. 3.—PLAN OF REDUCTION WORKS, ANDES COPPER MINING COMPANY.

system except the part used for the addition of fluxes is equipped with double conveyor belts. When only one grade of concentrate is being produced, one of the series of belts is stopped.

The average concentrate, including that going to the acid-plant roasters, has approximately the following analysis: copper, 29.17 per cent; sulfur, 32.47; iron, 22.13; insoluble, 13.86.

Sulfide Roaster Plant

There are two roaster plants; namely, the sulfide and acid-plant roasters. The acid-plant roasters treat dry concentrate from the sulfide roasters and produce the necessary sulfur dioxide gas for sulfuric acid generation, and calcine that is smelted in the reverberatory furnaces.

The sulfide roaster building, 38 ft. wide by 208 ft. long by 80 ft. high, is constructed of structural steel, sheathed and roofed with corrugated iron. Roasting equipment consists of seven Wedge roasters with their charge and discharge hoppers, Cottrell dust precipitators, service elevator, etc. The roasters in this plant can be used for calcining, but usually they are operated only as drying furnaces. They furnish the dry concentrate to the acid plant, which must be below 1 per cent moisture, and the dry fluxed concentrate going direct to the reverberatory, which averages approximately 3.5 per cent moisture.

The charge hoppers are filled by a series of conveyor belts, Nos. 19, 20 and 21, running from the flux bin through the concentrate filter plant to these hoppers. The hoppers have a capacity of 85 tons each and are discharged by 42-in. apron feeders. The furnaces are 22 ft. in diameter and have seven hearths and a drier hearth. When a roaster is being used for calcining concentrate, the hollow cast-iron arms are cooled with air under 10 oz. pressure. Oil burners for starting the calcining operation and for drying concentrate are on the third, fifth and seventh hearths. The firebrick muffles, or combustion chambers, in which the oil is burned, are 2 ft. long and have an opening 8 in. square. They are set into one of the furnace inspection doors on each hearth, as noted above. Fuel consumption for drying varies from 0.04 to 0.08 bbl. per ton of charge through the furnaces, depending on moisture content of feed and product. The roaster shaft makes 0.75 r.p.m. and the tonnage per roaster day varies from 90 to 160 tons according to dryness of feed and product. Each furnace is provided with a 25-ton discharge hopper into which the dried concentrate, or calcine, from the seventh hearth drops. Dust collected by the Cottrell dust precipitators falls vertically through iron pipes into these hoppers.

Each furnace is equipped with a two-compartment Cottrell treater of steel shell construction, which is above and to one side of the roaster. The two compartments operate independentlych o of eather, so that one

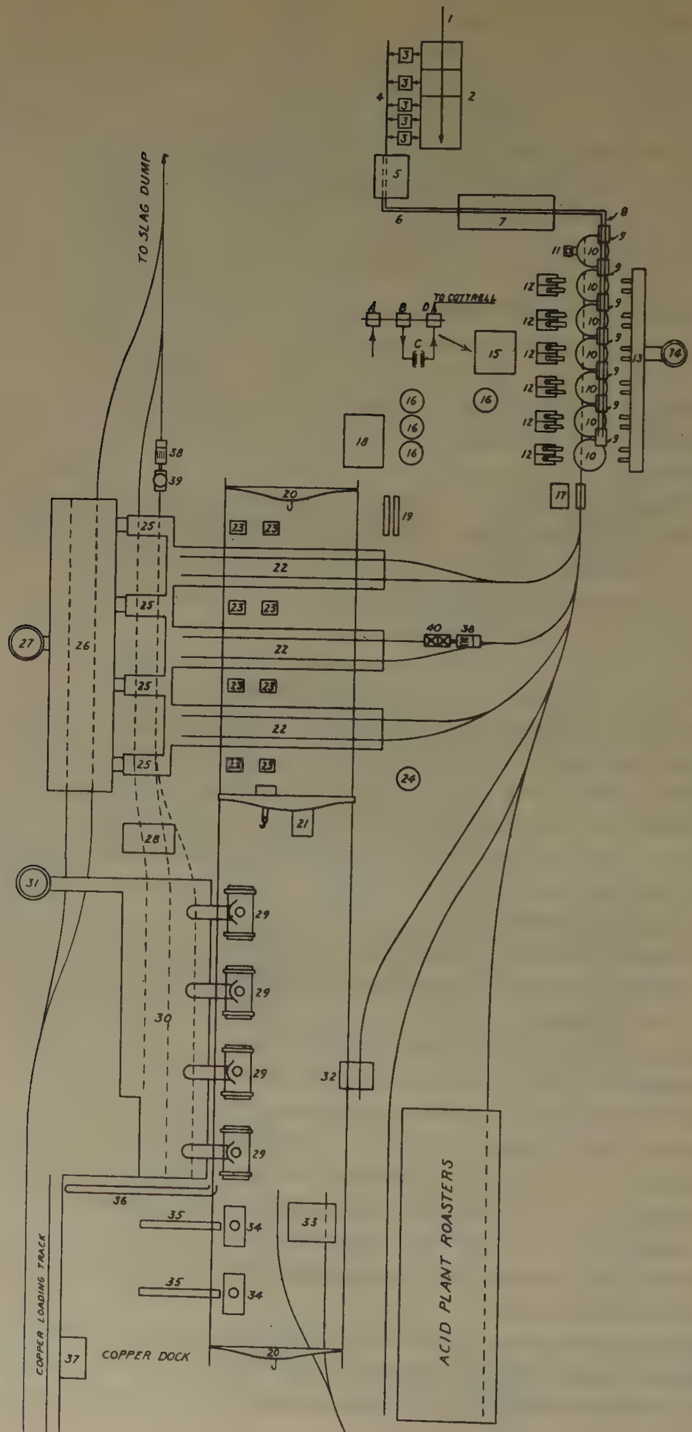


FIG. 4.—PLAN OF SMELTER. (DESCRIPTION ON OPPOSITE PAGE.)

1. Conveyor No. 18. A 14-in. belt from limerock crushing plant.
2. Flux bin. Wood construction. 20 by 20 × 72 ft. Divided into three compartments of 530, 340 and 680 tons capacity.
3. Five 18-in. apron feeders.
4. Conveyor No. 19. Double from filter plant, 14 in. by 300 ft., 335 ft. per min.
5. Concentrate filter plant.
6. Conveyor No. 20. Double belt, 14 in. by 185 ft., 335 f.p.m.
7. Concentrate storage bin. Wood construction. Capacity 1879 tons concentrate.
8. Conveyor No. 21. Double belt, 14 in. by 244 ft., 335 f.p.m. Rubber-edged angle scrapers used for removing material.
9. Seven charge hoppers. Capacity 85 tons each. Equipped with 42-in. apron feeders. Speed 6 ft. per min. Ratchet drive from roaster shaft.
10. Seven Wedge roasters. 22-ft. dia. Seven hearths and drier. 0.75 r.p.m. Air-cooled arms.
11. Seven wooden stacks. (All roasters have wooden stacks.)
12. Six plate and wire-type double-unit Cottrell treaters. Each unit 12 by 12 ft. by 28 ft. 9 in. Plate spacing, 11⅝ in. Wire, No. 14 hard wrought iron; plates, corrugated No. 14 hard wrought iron.
13. Header flue. Steel construction, 7 ft: 6 in. by 130 ft.
14. Steel stack. Brick-lined. 16 by 200 ft.
15. Cottrell substation. No. 8 Sturtevant blowers, 11,000 cu. ft. per min., 10 oz. static head, furnish air for cooling roaster arms.
    Three power units and three 75-kva. m.g. sets:
    A = a.c. squirrel-cage motor, three-phase, 60 cycles, 2200 volts.
    B = a.c. generator, single-phase, 60 cycles, variable voltage.
    C = 75-kva. transformer, single-phase, 60 cycles, 200-75,000 volts max. Taps for lower voltages.
    D = mechanical rectifier.
16. Oil-storage tanks: one roaster storage tank, capacity 16,000 gal.; three reverberatory storage tanks, total capacity, 36,000 gal.
17. Track scales, capacity 50 tons.
18. Roots blower room: one No. 6½ blower; two No. 5 blowers; three Quimby oil pumps.
19. Two steam oil heaters.
20. Two 5-ton repair cranes, one at each end of converter-reverberatory aisle above 60-ton cranes. Bridge speed, 75 f.p.m.; trolley speed, 50 f.p.m.; hook speed, 50 f.p.m.
21. Two 60-ton cranes, steel-mill type (extra heavy special design).
    Two 60-ton main hoists per crane, full load speed 10 f.p.m.
    One 10-ton auxiliary hoist per crane, full load speed 30 f.p.m.; bridge travel, full load speed 200 f.p.m.; trolley travel, full load speed 150 f.p.m.
22. Three reverberatories, 115 by 21 ft., dry bottom, slag 2 ft., silica 2 ft. Side charging. Charge tracks on long axis of furnace. Oil-fired, using 4-lb. air for atomizing oil preheated to 150° F. Oil pressure 50 lb. per sq. in.
23. Eight matte-ladle pits accommodate 115-cu. ft. matte ladles.
24. Clay mill, 9-in. wet pan.
25. Eight 400-hp. Stirling waste-heat boilers. 4088 sq. ft. effective heating surface per boiler. Working pressure, 400 lb. per sq. in. Two boilers in tandem per setting.
26. Reverberatory dust-chamber settling area. 8050-sq. ft. steel and hollow tile construction. Hopper bottom.
27. Steel stack. Brick-lined, 22 by 250 ft.
28. Boiler feed water pump room. Three steam turbine pumps.
29. Four 12 by 26-ft. Peirce-Smith basic-lined converters, Tacoma-type hood. Max. shell speed, 0.31 r.p.m. Equipped with automatic tilting device, protecting tuyeres in case of power failure; 38 tuyeres, 1½-in. dia.
30. Converter dust chamber. Settling area 7220 sq. ft. steel and hollow tile construction. Hopper bottom.
31. Steel stack, 22 by 200 ft. Unlined.
32. Bin for silica flux; 300-ton capacity.
33. Crushing plant for converter cleanings. 30-ton steel bin and grizzly; jaw crusher, Blake type, 15 by 24 in.; 39-bucket elevator, 12 in. by 82-ft. belt; 16 by 36-in. rolls.
34. Two receiving furnaces, 8 by 18 ft., oil-fired. Tilting. Speed 0.95 r.p.m. Magnesite brick lining.
35. Two straight-line casting machines. Speed 50.5 f.p.m. 42 molds per machine. Blister-copper molds, 18 by 32 in. by 3½ in. deep. Bar weighs approximately 300 lb.
36. Converter air line, 45-in. dia., 13 to 15 lb. air.
37. Blister-copper scales, capacity 6 tons.
38. Storage-battery locomotives: two 5-ton locomotives; five 15-ton locomotives.
39. Six slag-pot cars; capacity 145 cu. ft.
40. Eleven calcine cars; capacity 410 cu. ft.

side may be cut out and cleaned while the other is in use. Each double unit is 12 ft. square and 30 ft. high and is the ordinary plate and wire-type treater with gases passing through it vertically. The No. 14 corrugated, hard wrought-iron plates are spaced 11⅝ in. apart. No. 14 hard-drawn iron wires are used for the discharge electrodes. Current for the Cottrell units varies from 30,000 to 40,000 volts and is supplied by a special substation adjacent to the roaster plant. The steel shells are heat-insulated on the outside with plastered expanded metal, an air gap being left between the shell and the plaster.

The exit gases from the furnaces are disposed of in three ways. If the roasters are calcining, the gases pass through the Cottrell dust precipitators and an unlined steel header flue to a brick-lined steel stack, which is 16 ft. in diameter and 200 ft. high. When the roasters are producing dry concentrate (under 1 per cent moisture) for the acid plant, the gases pass from the Cottrell treaters through a short steel stack direct to the air. If concentrate is being dried for the reverberatory (3+ per cent moisture) practically no dust is produced, so the Cottrell dust precipitators are blanked off and the gases are discharged through short wooden stacks. The gases from the drying operation are relatively cool and as a result part of the moisture condenses in Cottrell treaters and flues, and corrosion follows. For this reason these units are bypassed when not needed.

The hoppers under the furnaces are discharged through self-cleaning slide gates into 410-cu. ft. hopper-bottomed calcine cars. After sampling and weighing, these cars are moved to the acid-plant roasters, or to the charge floor of the reverberatory plant, by 15-ton storage-battery locomotives.

Main Smelter Building and Cranes

Reverberatory furnaces and converters, together with the receiving and casting equipment, are contained in a structural steel building. The main section of the building is 75 ft. wide and 550 ft. long. Of this, 200 ft. at the north end is occupied by the reverberatory furnace and the remainder forms the converter aisle. The crane runway, with a span of 75 ft., extends the full length (550 ft.) of this section. At the north end of the building an extension 200 ft. long by 42 ft. wide encloses the firing end of the reverberatories and the oil heating equipment. This wing is on the east side of the main structure. On the opposite, or westerly side, a similar extension, 200 ft. long by 67 ft. wide, houses the front end of the reverberatory furnaces, waste-heat boilers and boiler-feed pumps. The locations of the different units are shown on the smelter plan. In studying this plan (Fig. 4) it is essential to bear in mind that the floor level is the same throughout this building.

Two 60-ton cranes serve both the reverberatories and converters. The crane rails are 50 ft. above the floor level. Each crane has two 60-ton main hoists and one 10-ton auxiliary hoist, which is between the two main hoists. At each end of the craneway, over the 60-ton crane at the end of its run, is a 5-ton single-hook repair crane. The 60-ton cranes transfer, by means of 115-cu. ft. cast-steel ladles, matte from the reverberatories to converters, copper from converters to receiving furnaces and molten converter slag from converters to reverberatories. Boats holding approximately 5 tons each are used for handling the cold material such as converter silica, cold dope, and so forth.

Reverberatory Plant

Three reverberatory furnaces with inside dimensions of 21 by 115 ft. are at the north end of the craneway. They extend across and are at right angles to the crane run. The back, or firing, walls are 16 ft. east of one crane rail and the skimming ends 24 ft. west of the other crane rail. The furnaces are spaced 50 ft. center to center with a distance of 22 ft. between buckstays on adjacent furnaces. The charge floors are 29 ft. below the crane rail. The furnaces are of the usual silica-brick construction except that, beginning 20 ft. from the burner wall, the insides of the side walls and the front wall of the furnace are lined with 12 in. of magnesite brick from the level of the silica bottom to somewhat above the slag line. For 10 ft. on each side of the matte-tapping blocks, on both sides of the furnace, the magnesite lining makes up the full thickness of the walls to the buckstays. The arch is horizontal for 57 ft. from the burner end and is 20 in. thick. From this point to the uptake, the arch is built of 15-in. silica brick and drops 2 ft. in this distance. The uptake extends the full width of the furnace and the connecting flues to the boilers are at each end of the uptake, at right angles to the longitudinal axis of the furnace.

The furnaces are side-charged from tracks paralleling the length of the furnace. The weighed charge is dropped directly from the 410-cu. ft. calcine cars into the furnace through 10-in. charge pipes. The molten converter slag is charged through a door in the side wall of the furnace, 10 ft. from the firing end and immediately under the skewbacks. This slag is poured by crane from the converter ladles into a cast-steel bowl, which is connected by a launder to a water-cooled, cast-iron runner passing through the side wall into the furnace.

One matte tap-hole is on each side of the furnace at a distance of 45 ft. from the front wall. There are eight matte ladle pits, 11 by 15 ft. by 8½ ft. deep, two on each side of each reverberatory furnace. The two pits between adjacent furnaces serve both units. The pits are situated to the right and left of each tap-hole, 7 ft. from the furnace wall,

and have a working space of 12 ft. between them. The matte is tapped into 115-cu. ft. cast-steel ladles which are in these pits. The filled ladles are hoisted by the crane to an elevation above that of the furnace charge floor and transferred by the same crane over the reverberatories to the converters.

Slag is skimmed intermittently through a door in the front wall of the furnace and flows through a port in the skimming floor into 145-cu. ft. slag pots, which are hauled to the slag dump by 15-ton storage-battery locomotives. The slag pots are tilted by 7½-hp. d.c. motors with controls on each car. The current for operating the tilting motors is furnished by the locomotive batteries.

Fuel oil is used for firing, and from four to nine burners of a modified Cananea type are used per furnace, according to the tonnage being treated. The oil is delivered to the burners at 150° F. and 50 lb. pressure. The oil is heated by steam and the pressure maintained by Quimby screw pumps. Atomizing air, supplied by Roots blowers, is varied in pressure from 2½ to 4½ lb. to suit combustion conditions in the furnace. One No. 6½ and two No. 5 blowers furnish air for the roaster as well as for the reverberatory burners.

The waste-heat boiler plant consists of four units, arranged so that two can be used by each furnace. A unit consists of two 400-hp. Stirling waste-heat boilers of 400 lb. working pressure, placed in tandem. The flues from the reverberatories enter the boiler setting at right angles to the long axis of the boilers. These short flues have a 40-sq. ft. cross-section and are provided with water-cooled dampers so that any boiler unit not in use can be cut off completely from the reverberatory. The slag and dust particles that settle in the forebay of the boiler are raked out through doors provided for that purpose. Brick-lined steel hoppers catch the dust that accumulates under the boilers, and these cleanings are returned to the reverberatory.

From the waste-heat boilers, the gases pass through short flues to the reverberatory dust chamber. Each flue has two butterfly dampers for draft control. The dust chamber, which has a settling area of 175 by 26 ft. and a height of 24 ft., is constructed of steel and hollow tile and has hoppers in the bottom for dust removal. From the dust chambers, the gases pass through a short flue to a brick-lined steel stack, 22 ft. in diameter and 250 ft. high.

Jones-Wraith draft regulators acting on the butterfly dampers keep the furnace draft constant to within 0.01 in. of water. An indicating, recording and integrating Brown flowmeter, checked by tank measurements, measures the oil to the furnace. A recording thermometer for oil temperature is used, and also a gage for recording pressure of atomizing air. Gas analyses are made at intervals to check the combustion.

Converter Plant

The converter plant occupies the south portion of the smelter building and consists of four Peirce-Smith converters lined with magnesite brick, two receiving furnaces, two casting machines and the slag crusher. The converter aisle is 350 ft. long by 75 ft. wide. The converters and receiving furnaces are on the west side of the aisle, with their longitudinal axis parallel to the crane run.

The converters are 12 by 26 ft. and have thirty-eight 1½-in. tuyeres. With matte containing 40 per cent copper, each converter can make approximately 90 tons of blister copper per converter day. Converter air at 13 to 15 lb. pressure is supplied by three 48 by 48 by 36-in. heavy duty duplex blowing engines, which are in the smelter power house. The converter gases pass through a Tacoma-type hood and gooseneck into the converter dust chamber, which has a settling area of 47 by 161 ft. and is 24 ft. high. It is constructed of steel and hollow tile, and contains hoppers for the removal of dust. A short flue connects the dust chamber to an unlined steel stack, 22 ft. in diameter and 200 ft. high.

The two oil-fired receiving furnaces are 8 ft. in diameter by 18 ft. long and are lined with 9 in. of magnesite brick. They receive the molten blister copper from the converters, and serve two link-belt, or straight-line casting machines, which are set at right angles to the converter aisle and extend outside of the building on to the copper dock. For casting blister cakes, each machine carries 42 molds, which are cast without splash plates from high-sulfur blister. The cakes are 18 by 32 in. by approximately 3 in. thick and average about 300 lb. each. The cakes from the casting machine are trimmed, weighed and loaded on flat cars for shipment. When casting anodes for production of starting sheets, the 42 blister-cake molds are replaced by 14 anode molds.

The slag crusher is in the south end of the converter aisle, adjoining the copper-receiving furnaces. Material to be crushed is dumped upon cast-steel grates having openings 9 in. square, where oversize material is broken by hand into the hopper-bottomed bin below. From this bin it is fed into a 15 by 24-in. jaw crusher set at 1½ to 2 in. The crusher discharges into the boot of an elevator, which in turn feeds a set of 36 by 16-in. crushing rolls. The rolls crush to approximately 1 in. and discharge into cars.

The flux bin for siliceous ore immediately adjoins the converter building opposite the row of four converters, and has a capacity of 300 tons. Siliceous ore is delivered to this bin in bottom-dump ore cars and is discharged, by means of chutes, into boats handled by the converter cranes. Siliceous ore used is the regular sulfide ore that has passed through the gyratory crushers and is loaded into cars in the fine-crushing

plant. It contains approximately 70 per cent silica and 16 per cent alumina.

The converters have an emergency automatic tilting device, which, by means of auxiliary storage-battery power, turns the converters out of the stack in the event of failure of power. This automatic device is non-acting except: (1) when the power goes off the line leading to the converter tilting motor, and (2) when the air pressure on the tuyeres falls below 7 lb. In either of these cases the power from the storage battery turns the converter enough to uncover the tuyere line.

Eleven 410-cu. ft. hopper-bottomed steel cars, five 15-ton and two 5½-ton storage-battery locomotives, all of meter gage, are provided for smelter service. Six of these cars have closed tops and are used for handling calcine and dry concentrate. The other five, which handle moist concentrate, secondaries, etc. have open tops. They are discharged through hand-operated sliding gates, which are enclosed in dustproof housings. The storage-battery locomotives are used for light switching throughout the reduction plant.

The data in Table 1 are typical of reverberatory smelting operations when calcining concentrates for sulfuric acid production.

TABLE 1.—*Data for Second Six Months of the Year* 1931

	Per Cent of Solid Charge
Siliceous fettling ore	0.15
Dried concentrates	20.90
Calcined concentrates	63.20
Smelter secondaries	12.80
Oxide-plant secondaries	1.20
Limerock in concentrates	1.75
	100.00

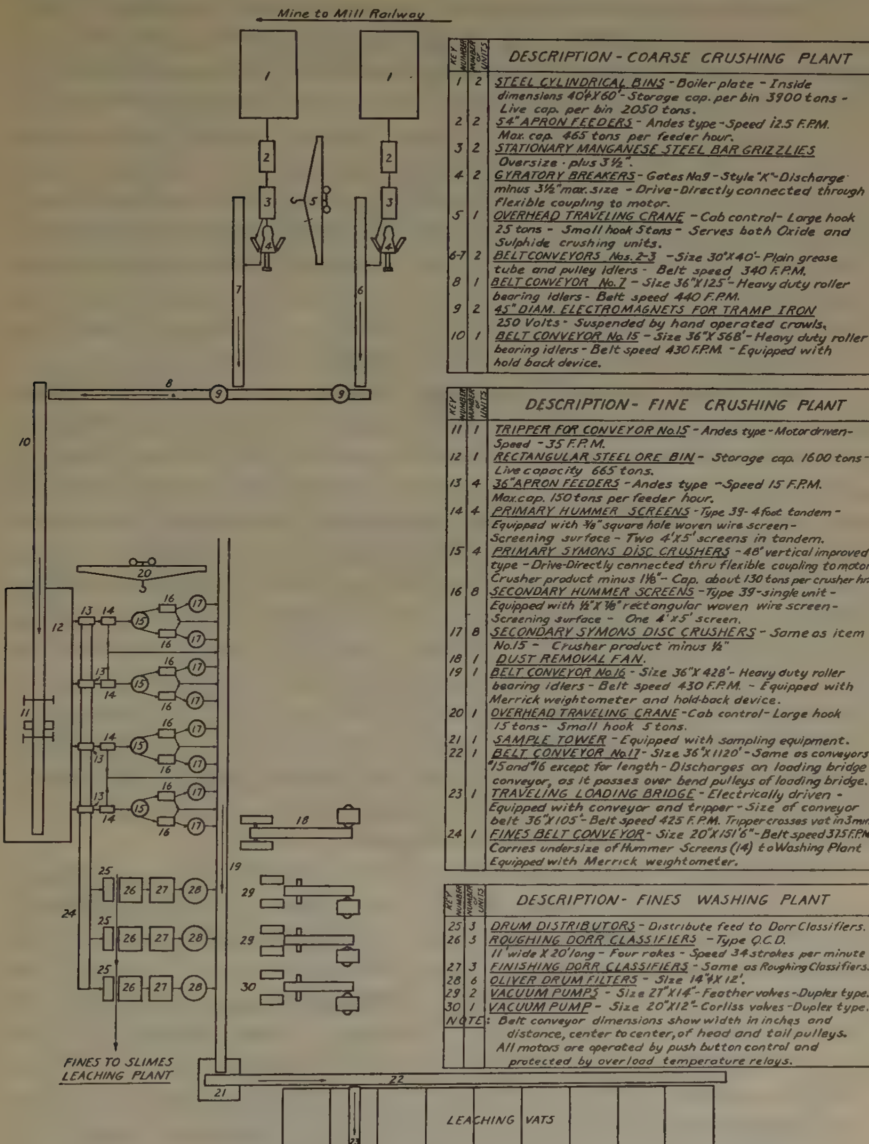
	Cu	S	SiO_2	FeO	Al_2O_3	CaO
Analysis of charge, per cent	21.33	18.	16.4	35.9	4.2	1.5
Analysis of slag, per cent	0.60	0.8	30.9	50.9	7.2	2.5
Copper in matte, per cent	45.88					

Fuel-oil ratio, bbl. per ton solid charge, 0.759.

COARSE-CRUSHING PLANT

The elevation of the mine railway was the controlling factor in fixing the location of the coarse-crushing plant and receiving bins. A narrow gulch that cuts through the upper part of the inclined shelf or tableland, forming the main plant site, provided a convenient location. A double-track steel trestle spans this gulch. The crusher building is

in the bottom of the depression and the railway crosses directly over the receiving bins.



KEY NUMBER	NUMBER OF UNITS	DESCRIPTION - COARSE CRUSHING PLANT
1	2	STEEL CYLINDRICAL BINS - Boiler plate - Inside dimensions 40'⌀X60' - Storage cap. per bin 3900 tons - Live cap. per bin 2050 tons.
2	2	54" APRON FEEDERS - Andes type - Speed 12.5 F.P.M. Max. cap. 465 tons per feeder hour.
3	2	STATIONARY MANGANESE STEEL BAR GRIZZLIES Oversize - plus 3½".
4	2	GYRATORY BREAKERS - Gates No. 9 - Style "K" - Discharge minus 3½" max. size - Drive - Directly connected through flexible coupling to motor.
5	1	OVERHEAD TRAVELING CRANE - Cab control - Large hook 25 tons - Small hook 5 tons - Serves both Oxide and Sulphide crushing units.
6-7	2	BELT CONVEYORS Nos. 2-3 - Size 30"X40' - Plain grease tube and pulley idlers - Belt speed 340 F.P.M.
8	1	BELT CONVEYOR No. 7 - Size 36"X125' - Heavy duty roller bearing idlers - Belt speed 440 F.P.M.
9	2	45" DIAM. ELECTROMAGNETS FOR TRAMP IRON 250 Volts - Suspended by hand operated crawls.
10	1	BELT CONVEYOR No. 15 - Size 36"X568' - Heavy duty roller bearing idlers - Belt speed 430 F.P.M. - Equipped with hold back device.

KEY NUMBER	NUMBER OF UNITS	DESCRIPTION - FINE CRUSHING PLANT
11	1	TRIPPER FOR CONVEYOR No. 15 - Andes type - Motor driven - Speed - 35 F.P.M.
12	1	RECTANGULAR STEEL ORE BIN - Storage cap. 1600 tons - Live capacity 665 tons.
13	4	36" APRON FEEDERS - Andes type - Speed 15 F.P.M. Max. cap. 150 tons per feeder hour.
14	4	PRIMARY HUMMER SCREENS - Type 39 - 4 foot tandem - Equipped with ⅜" square hole woven wire screen - Screening surface - Two 4'X5' screens in tandem.
15	4	PRIMARY SYMONS DISC CRUSHERS - 48" vertical improved type - Drive - Directly connected thru flexible coupling to motor. Crusher product minus 1⅛" - Cap. about 130 tons per crusher hr.
16	8	SECONDARY HUMMER SCREENS - Type 39 - single unit - Equipped with ½"X⅛" rectangular woven wire screen - Screening surface - One 4'x5' screen.
17	8	SECONDARY SYMONS DISC CRUSHERS - Same as item No. 15 - Crusher product minus ½"
18	1	DUST REMOVAL FAN.
19	1	BELT CONVEYOR No. 16 - Size 36"X428' - Heavy duty roller bearing idlers - Belt speed 430 F.P.M. - Equipped with Merrick weightometer and hold-back device.
20	1	OVERHEAD TRAVELING CRANE - Cab control - Large hook 15 tons - Small hook 5 tons.
21	1	SAMPLE TOWER - Equipped with sampling equipment.
22	1	BELT CONVEYOR No. 17 - Size 36"X1120' - Same as conveyors #15 and #16 except for length - Discharges on loading bridge conveyor, as it passes over bend pulleys of loading bridge.
23	1	TRAVELING LOADING BRIDGE - Electrically driven - Equipped with conveyor and tripper - Size of conveyor belt 36"X105' - Belt speed 425 F.P.M. Tripper crosses vat in 3 min.
24	1	FINES BELT CONVEYOR - Size 20"X151'6" - Belt speed 375 F.P.M. Carries undersize of Hummer Screens (14) to Washing Plant Equipped with Merrick weightometer.

KEY NUMBER	NUMBER OF UNITS	DESCRIPTION - FINES WASHING PLANT
25	3	DRUM DISTRIBUTORS - Distribute feed to Dorr Classifiers.
26	3	ROUGHING DORR CLASSIFIERS - Type Q.C.D. 11' wide X 20' long - Four rakes - Speed 34 strokes per minute
27	3	FINISHING DORR CLASSIFIERS - Same as Roughing Classifiers.
28	6	OLIVER DRUM FILTERS - Size 14'⌀X12'.
29	2	VACUUM PUMPS - Size 27"X14" - Feather valves - Duplex type.
30	1	VACUUM PUMP - Size 20"X12" - Corliss valves - Duplex type.
NOTE:		Belt conveyor dimensions show width in inches and distance, center to center, of head and tail pulleys. All motors are operated by push button control and protected by overload temperature relays.

Fig. 5.—Layout of oxide crushing and fines-washing plants.

The installation includes five cylindrical steel ore bins, an equal number of No. 9-K Gates gyratory crushers, and the necessary auxiliary equipment such as apron feeders, conveyors, magnets, etc. The crushing machinery is enclosed in a structural steel and concrete building, 56 ft.

wide by 268 ft. long. The roof and sides of the structure are covered with corrugated iron.

The ore is received from the mine in trains of 45-ton Ingoldsby bottom-dump ore cars. Just before the trains reach the crushing-plant ore bins, they pass over a Fairbanks track scale which is equipped with a Streeter-Amet tape-printing recording device.

The coarse-crushing plant is made up of five identical parallel units, the three northern units handling sulfide ore and the two southern units handling oxide ore. Each crushing unit is provided with a steel cylindrical storage bin, 40 ft. in diameter and 60 ft. high, which receives run-of-mine ore (through mine grizzlies with 10-in. spacing). These bins are above and behind the crushers and are carried on massive concrete foundations. They are just far enough apart to give clearance for structural steel legs that support the railroad trestle over the bins.

Ore is drawn through a rectangular opening in the center of the bin floor by a 54-in. apron feeder of Andes design. The ore is moved about 15 ft. forward, where it falls upon an ore-bedded shelf to retard velocity, and passes over two sets of stationary manganese-steel grizzly bars, arranged in tandem. All spill from undersize of feeder falls into undersize chutes of grizzlies by gravity so that cleanup is negligible. Grizzly bars are spaced at 2¼ in. at upper and 2¾ in. at lower end. One-half to two-thirds of the ore passes through the grizzlies as undersize, and the oversize feeds into a gyratory crusher. Scrap mine timbers and heavy tramp steel are removed by hand before the ore drops on the grizzlies.

The crushing plant is served by a cab-operated, four-motor, over-head traveling crane, equipped with hooks of 25 and 5 tons capacity, which travels the length of the building over the gyratory crushers and feeder-floor galleries. At the south end of the building there is a spare bay for repair work served by a railroad spur track.

Crushers are set so that maximum size of crushed ore does not exceed 3½ in. At this setting, crushers take full capacity of feeders, which is about 500 tons of sulfide ore per hour. Manganese-steel concave consumption is approximately 1.5 lb. per 1000 tons ore.

The discharge of grizzly undersize chute and gyratory crusher go to a short inclined 30-in. belt conveyor. The grizzly undersize is fed on to the conveyor belt first and is followed by crusher discharge.

The coarse-crushing plant breaks the run-of-mine oxide ore (through mine grizzlies with 10-in. spacing) to about 3.5 in. maximum size.

Two 30-in. conveyors of the two oxide units deliver to a 36-in. conveyor (No. 7) running at right angles to them and to the south. Two 45-in. magnets overhang this belt following the discharge chutes of conveyors Nos. 2 and 3.

Inclined conveyor belt No. 15, which is 36 in. wide and 585 ft. long, receives the ore from collecting conveyor No. 7, in the coarse-crushing

plant, and delivers it to the receiving bin of the oxide fine-crushing plant, distribution over the length of this bin being effected by means of a motor-driven tripper.

When operating the units at capacity, the power consumption in the coarse-crushing plant, excluding conveyor No. 15, follows: Crushing, 0.0779 kw-hr. per ton ore through plant; conveying, 0.0348; lighting, 0.0144; total 0.1271.

Oxide Metallurgical Plant

The oxide ores as delivered from the mine are crushed in the coarse-crushing plant that crushes both sulfide and oxide ores.

Fine-crushing Plant

The oxide fine-crushing plant crushes broken ore from the coarse crushers to minus ½-in. maximum size. The plant is contained in a structural steel building, 70 ft. wide by 137 ft. long. The roof and sides of the building are covered with corrugated iron. There is an extra bay at the east end of the building for repair work and for entrance of a railroad spur track. The crushers are served by a cab-controlled, four-motor, overhead traveling crane with hooks of 5 and 15 tons capacity.

The ore is crushed in two stages from 3½ to ½ in. maximum size. There are four parallel crushing sections, each consisting of one primary and two secondary Symons disk crushers, and auxiliary equipment.

The fine-crusher receiving bin serves the four crushing sections. The bin is of steel construction, 100 ft. long by 19 ft. wide by 18 ft. deep, with a flat bottom, and has an ore-storage capacity of 1600 tons. Ore is drawn from the receiving bin by four apron feeders, through openings in the bin bottom.

The feeder discharge passes over a type 39, 4 by 5-ft. tandem Hum-mer screen equipped with Ton-cap screens having ⅜-in. square openings. The undersize, which varies from 33 to 40 per cent of the total ore, drops upon a 20-in. conveyor belt and is delivered to the washing plant. The treatment of this material will be covered in the description of the washing plant. The screen oversize forms the feed to the primary Symons crushers.

The Symons crushers in the oxide fine-crushing plant are identical with those in the sulfide fine-crushing section. The primary Symons crushers are set to crush to approximately 1⅛ in. The discharge from each of these crushers is halved by keeping the products of the two discharge chutes separated. Each of these halves furnishes the feed to a type 39, 4-ft. Hum-mer screen equipped with woven wire screen having ½ by ⅞-in. rectangular openings. The oversize of each screen forms the feed to a secondary Symons crusher and the undersize joins the secondary crusher product.

The secondary Symons crushers are set to give a product with a maximum size of ½ in. The secondary crusher product, along with the secondary screen undersize, falls through vertical steel chutes on conveyor No. 16, which is in a tunnel under and in line with the fine crushers.

The gross steel consumption in Symons disk crushers in the oxide fine-crushing plant has averaged 11.18 lb. per 1000 tons of ore.

The only place in the crushing division where a dust problem is encountered is in the oxide fine-crushing plant on the secondary crusher floor, where there is considerable dust resulting from fine crushing, screening and the dropping of the ore on to conveyor No. 16. It is

FIG. 6.—OXIDE FINE-CRUSHING PLANT; FEEDERS, PRIMARY SCREENS AND PRIMARY SYMONS.

removed by a fan, which sucks the dust-laden air through a header that has dust ducts at each loading chute in No. 16 conveyor tunnel. Before reaching the fan, the dust enters a cyclone dust collector, which removes most of the solids. These solids are transferred to the washing plant for treatment.

Conveyor No. 16, which is 36 in. wide and 428 ft. long, conveys the ore from the fine-crushing and washing plants to the sample tower. This conveyor is horizontal in the tunnel under the fine-crushing and washing plants and is then inclined upward to the sample tower. At the center of the incline, the conveyor belt passes over a Merrick weightometer.

Conveyor No. 17 is 36 in. wide. It receives its load from the transfer chute in the sample tower and runs to the north, a distance of 1120 ft., along the east side of the leaching vats. When the conveyor belt reaches

the loading bridge, it passes up a short inclined conveyor way and over two bend pulleys on the loading bridge, and the ore falls into the chute which feeds the loading-bridge conveyor.

The sample tower is a five-story steel building at the junction of conveyors 16 and 17, and contains the sampling equipment. A sample of the ore stream is taken by a bucket sampler as the ore falls from conveyor 16. The sample is elevated and dumped into a small hopper, which feeds a set of crushing rolls (10 in. long by 18-in. diameter). The roll product is cut down by means of a slotted belt sampler. The reject from this belt is delivered to conveyor 17, and the sample, which weighs about 1000 lb. per charge, is transferred to the laboratory sample mill.

All conveyors and feeders in the oxide-crushing plant are so tied in by electric governing devices that stoppage of a conveyor belt stops preceding conveyors and feeders.

The power consumption, starting at conveyor No. 15 and ending with ore delivered in vats, is as follows: crushing, 0.643 kw-hr. per ton of ore; conveying, 0.286; washing plant, 0.242; sample tower, 0.019; loading bridge, 0.029; lighting, 0.031; total, 1.250.

Oxide Washing Plant

Experimental leaching work on the Andes oxide ore in a pilot plant demonstrated that the fines in the crushed ore were detrimental to leaching. When the test unit was started erratic and unsatisfactory results were recorded. Later the practice was established of adding water to the crushed ore before charging it into the leaching vats, until it contained approximately 4.5 per cent moisture. Uniform and satisfactory leaching resulted from this practice, therefore when the leaching plant was built provisions were made for moistening the ore before charging it into the vats. Soon after work at the leaching plant was started it became evident that merely moistening the ore before charging would not give satisfactory leaching results. An increased percentage of fines in run-of-mine ore, as well as variations in quantity, caused unsurmountable difficulties in the bedding of uniform and permeable charges. Several different methods of wetting and charging were tried, as well as both upward and downward percolation. None of these improved the extraction appreciably.

Tests proved that the ore would leach satisfactorily provided a sufficient proportion of fines had been removed, and in order to remove the necessary amount, and the most harmful portion—i. e., the primary slimes—the washing plant was constructed.

The washing units are enclosed in a three-story building which adjoins the oxide fine-crusher building on the west.

The primary Hum-mer screen undersize (−⅜ in.) from the fine-crushing plant is delivered to the washing plant by a 20-in. conveyor belt which is equipped with a Merrick weightometer. There are three

washing and dewatering units, each made up of two Dorr classifiers, two Oliver filters and the necessary auxiliary equipment. Scrapers, which serve as splitters, are installed over the 20-in. belt in such a manner that the material to be washed can be delivered to any one or all of the washing units. Usually it is not necessary to wash all of the $-3/8$-in. product; but, owing to the dampness of the ore and the tendency for the screens to blind, it is not feasible to use screens of a finer mesh. The portion that is not to be washed is scraped off the belt ahead of the washing units and falls into a steep chute, where it is moistened with water and delivered to conveyor 16.

The two Dorr classifiers of each unit are 11 ft. wide by 20 ft. long, and operate in tandem. The cut from the 20-in. belt at each unit is sluiced into a drum distributor, which feeds the rougher classifier. The sand from the rougher classifier is advanced into the finishing classifier. The washed sand from the finishing classifier is discharged to an Oliver filter.

The feed to the rougher classifier averages 18 to 20 per cent minus 200 mesh. The overflow from the rougher classifier is laundered to the slimes-leaching plant. It contains about 10 per cent solids and amounts to approximately 5.25 per cent of the ore fed to the fine-crushing plant. An average cumulative screen analysis of these slimes is: +48-mesh, 0.30 per cent; +65-mesh, 1.43; +100-mesh, 6.36; +150-mesh, 14.34; +200-mesh, 27.40; −200-mesh, 72.60.

The overflow from the finishing classifier is elevated by an air-lift and used for sluicing the sands into the rougher classifier. The washed sand that forms the feed to the Oliver filter contains approximately 2 per cent minus 200-mesh material.

Two Oliver filters are provided for each unit but only one is used at a time, the other serving as a spare. Five of the filters are the usual drum type, except that they have two automatic valves, one on each trunnion. They are 14-ft. diameter by 12 ft. long. The sixth is a specially designed Oliver tray filter. Coco matting is used for covering the filters. The vacuum is maintained at 2 in. of mercury. A higher vacuum tends to blind the coco matting. Under normal operating conditions the cake averages about 3 in. in thickness. The discharge from the filters, which contains approximately 11 per cent moisture, falls directly upon conveyor 16 on top of the crushed dry ore. The dry ore absorbs enough moisture so that the conveyor belt does not become wet or caked with fines.

A close control of the minus 200-mesh material in the charge to the vats is essential. In general, enough of the fines are washed to hold the minus 200-mesh at not over 6.5 per cent. An average cumulative screen analysis of the vat charge is as follows: +0.525-in., 2.96 per cent; +0.371-in., 19.33; +4-mesh, 51.60; +8-mesh, 67.09; +20-mesh, 80.08; +48-mesh, 87.91; +200-mesh, 93.99; −200-mesh, 6.01.

Representative fine-crushing and washing-plant data for a 7500-ton vat charge are given in Table 2.

TABLE 2.—*Data for 7500-ton Vat Charge*

Ore through fine-crushing plant, tons	7,908
Ore washed, tons	1,802
Fines to slimes-leaching plant, tons	408
Washed sand to vat, tons	1,394
Unwashed ore to vat, tons	6,106
All ore to vat, tons	7,500
Ore washed, per cent of total	22.79
Ore to slimes-leaching, per cent of total	5.16
Washed sands, per cent of vat charge	18.59
Washed sands, per cent moisture	11.52
Total charge to vat, per cent moisture	6.25

LEACHING PLANT

The leaching plant was designed to treat 7500 tons of ore per day by a six-day counter-current sulfuric acid leach, followed by the usual counter-current washing.

The plant consists of a single row of nine leaching vats, a parallel row of eight solution sumps, a loading bridge, an unloading bridge, pumps and pipe lines. The sumps are elevated to permit a gravity flow of solution from the bottom of the sumps over the top of the vat walls. The solutions from the vats flow by gravity to the pump house, which is at the south end of the solution sumps, and are elevated to the solution sumps by horizontal centrifugal lead pumps. Lead pipe lines of suitable dimensions are provided for the transfer of all solutions. The ore is delivered to the loading bridge by a 36-in. belt conveyor (No. 17), which is on the upper side and extends the full length of the row of vats. The unloading bridge excavates and delivers the tailing to the sluicing system.

The leaching vats are of reinforced concrete and are supported on concrete piers, 6 ft. 3 in. high. They are built in four sets of two vats each and a single vat. The inside dimensions of each vat are: length, 105 ft.; width, 115 ft. and depth 19 ft. 6 in. The floors are lined with 1½-in. of mastic and the walls with 8½-in. of acidproof brick laid up in mastic mortar. The filter bottoms are of the Inspiration type; 6 by 6-in. sills, spaced on 16-in. centers, rest on the mastic floor. On top and at right angles to the sills, 2 by 3-in. boards are placed at 12-in. centers. The filter boards, which form a continuous floor, are laid on top and at right angles to the 2 by 3-in. boards. The filter boards are 2 by 12-in. planks with ship-lap joints and have thirty-six ⅜-in. holes per square foot of surface, countersunk with a ¾-in. hole from beneath. Stringers, 4 by 6-in., spaced at 30-in. centers, rest on the filter boards. They serve as wearing skids for the unloading bridge bucket.

Six 12-in. outlet connections are in the bottom of each vat, three at each end. Each outlet is connected to a 10-in. air-lift. The center outlet on the east end is used also for draining solution from the vat. Four of the air-lifts are used for circulation and two for advancing solution from the bottom of one vat to the top of the next. They discharge over the vat walls and have a capacity of 250 tons of solution per hour per lift. Air for these lifts is supplied by compressors in the purification plant. The air pressure varies from 20 to 25 pounds.

The construction of the concrete solution sumps is similar to that of the leaching vats. There are two blocks of four tanks each. Each sump in the first block is 52 by 89 ft. by 18 ft. deep, and has a capacity of 2500 volumetric tons of solution. Two of these are lined with acid-proof brick, like the leaching vats, and are used for the storage of strong solution. The other two are lined with 10-lb. sheet lead and are utilized for storing spent electrolyte. The second block of sumps is used for the storage of wash waters and these sumps are lead lined. Two are 38 by 89 ft. by 18 ft. deep, and the other two are 42 by 89 ft. by 18 ft. deep. Each sump has a single outlet in the bottom.

There is a set of lead pipe lines above and below both the leaching vats and the solution sumps, connected in such a way that solutions can be transferred from the bottoms of the solution sumps to the tops of the vats, and from the bottoms of the vats to the pump house, from which they can be elevated to the solution sumps. Interconnections make it possible to transfer solutions in almost any manner desired.

The lead pipe lines, larger than 4 in. in diameter, are all of the same construction except that the thickness of the lead is varied to conform to the different solution heads. The pipes are lagged on the outside with 1-in. square wooden strips bound with strap-iron hoops, and rest in wooden saddles. Ajo-type rubber-diaphragm lead valves are used.

The loading bridge that spans the leaching vats is of structural steel. It is mounted on power-driven trucks and carries a motor-driven belt conveyor and a tripper, which is actuated by a cable attached to an electric winch. The ore from conveyor 17 falls on to the bridge conveyor, from which it is discharged by the tripper. The ore from both sides of the tripper drops into a V-shaped hopper, which runs the full length of the bridge. In this hopper the two streams of ore are brought together and discharged through a narrow slot into the leaching vat.

The unloading bridge for excavating the tailing is of structural steel. It spans the loading bridge as well as the leaching vats. It is mounted on power-driven trucks and is provided with the necessary equipment for the operation of an 8-ton clamshell digging bucket. The bucket dumps the tailing into a 50-ton hopper built in one leg of the bridge. A 40-in. belt feeder discharges the tailing from the hopper into a concrete sluicing launder, on the lower side and parallel to the row of vats. The

tailing is sluiced from this launder into the main tailing launders, which deliver to the oxide-tailing dams. Sluicing is done with unsettled concentrator tailing and water. Drag-line excavators are used for maintaining the dams.

The oxide ore treated in the leaching plant is found mostly in mineralized and altered diorite porphyry, although some ore extends into the limy and shaly sediments into which the porphyry has been intruded. The important oxidized copper minerals are malachite, azurite and cuprite, named in the order of abundance.

One leaching vat is charged, one charge is covered with solution, and one vat excavated for each day of 24 hr. Nine days are required to complete the leaching cycle, divided as follows: charging ore to vat, 1 day; leaching, 6 days; washing, 1 day; excavating, 1 day.

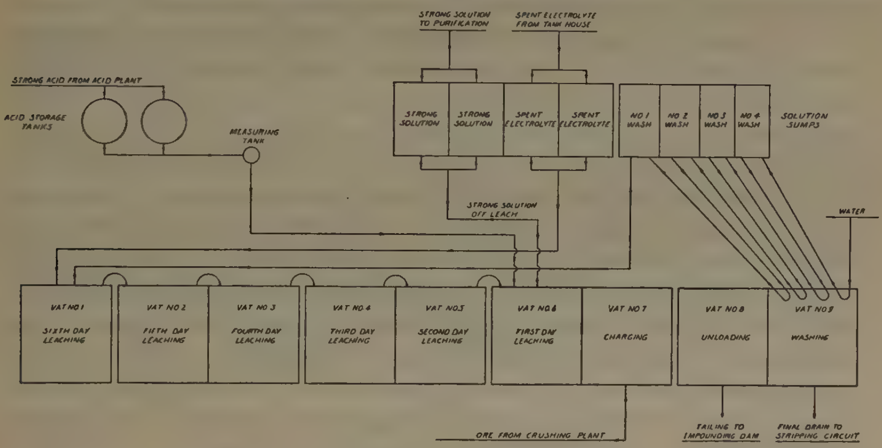


FIG. 7.—FLOW SHEET OF LEACHING PLANT.

Normally about 15 hr. is required for charging 7500 tons of ore into the vat. The ore is not bedded. In charging a vat, it is the practice to fill along a side wall to the height to which the vat is to be charged. Thereafter, at the end of each round trip of the loading-bridge tripper, the loading bridge is moved forward 12 to 15 in. In this way the filled portion of the vat is carried at the final level. The ore column is approximately 17 ft. high.

The ore is leached for six days by the counter-current system. Downward percolation is used except when a new charge of ore is being covered. Upward percolation is used in covering. Six charges are leaching simultaneously with the solution on each charge circulating at approximately 2.5 tons per 24 hr. per ton of ore, and also advancing from vat to vat at about 0.70 tons per ton of ore per 24 hours.

The flow sheet and cycle of leaching plant operations (Fig. 7) for a given day indicate the leaching process in detail. On the first day of

leaching, the new charge of ore is covered by upward percolation with advance solution from the preceding charge, which is low in acid and high in copper. In order to prevent neutralization and avoid the precipitation of ferric salts in the charge, the required amount of 60° Bé. sulfuric acid is added as rapidly as possible to the advance solution during the covering period.

As soon as the ore is covered the advances are stopped and the solution on the fresh charge circulated until the acid content of this solution has been reduced to around 10 grams per liter. The withdrawal of the enriched solution is then started from the bottom of the vat and continues until the predetermined volume is drained. This volume is calculated from the known copper content of the charge. This enriched solution which, after purification, is sent to the electrolytic plant, is known as the "strong solution." It contains approximately 41 grams per liter copper, 14 grams per liter H_2SO_4, and 7.6 grams per liter iron. The advances from vat to vat are resumed about one hour after the withdrawal of strong solution is started. On a given charge on the succeeding days there is a daily increase in acidity and decrease in copper content of the solution on the charge. On the sixth day the required volume of spent electrolyte solution from the tank house is added, containing approximately 11 grams per liter Cu, 76 grams per liter H_2SO_4, and 1.75 grams per liter Fe. It is followed by the No. 1 wash water. On the seventh day the solution advance is stopped and the remaining three wash waters are put on successively, displacement washing being used. The last wash is followed by a definite volume of fresh water, usually amounting to about 36 per cent of the weight of the ore. The charge is allowed to drain, and on the eighth day the tailings are excavated. This requires about 16 hours.

A portion of the fourth wash off the charge and the final drain from the tailings are discarded from the leaching circuit each day. The discarded portion of the No. 4 wash goes to the slimes-leaching cementation launders and the final drain to the stripping circuit in the purification plant. These discards are made for controlling solution volumes and not as a means of bleeding impurities.

Appreciable amounts of iron and aluminum, and smaller amounts of arsenic, phosphorous, chlorine, etc., are dissolved in the leaching operation. About 82 per cent of the iron dissolved is in the ferric state. For the year 1931 the average iron and aluminum solubilities were:

	Pounds
Iron dissolved per pound copper extracted	0.202
Ferric iron dissolved per pound copper extracted	0.165
Aluminum dissolved per pound copper extracted	0.066

The impurity content of the solutions is controlled by purifying all strong solution from the leaching plant. In the purification plant,

ground limerock is used for precipitating iron, aluminum, phosphorous and arsenic. The chlorine is removed from the purified strong solution by cement copper.

The acid consumption varies with the nature and grade of the ore. Average acid figures for the year 1931 were: new 60° Bé. acid used per ton ore, 64.48 lb.; 100 per cent H_2SO_4 neutralized per ton ore, 62.30 lb. Leaching plant data for the year 1931 are given in Table 3.

TABLE 3.—*Leaching Plant Data for the Year* 1931

	To Leaching		From Leaching		Extraction
	Ore	Spent Electrolyte	Tailing	Strong Solution	
Moisture, per cent	6.27		12.49		
Screen analysis:					
Per cent on 4 mesh	51.60				
Per cent on −20 mesh	19.92				
Per cent on −200 mesh	6.01				
Total copper, per cent	1.370		0.145		89.696
Acid-soluble copper, per cent			0.102		
Water-soluble copper, per cent			0.015		
Oxide copper, per cent	1.308				91.309
Sulfide copper, per cent	0.062		0.028		
Copper dissolved per ton ore, lb					24.573
Grams per liter:					
Copper		10.71		41.55	
H_2SO_4		75.95		14.11	
Total iron		1.73		7.03	
Ferric iron		1.08		6.32	
Aluminum		4.10		4.99	
Chlorine		0.14		0.29	
HNO_3		0.07		0.085	
Arsenic		0.26		1.40	
P_2O_5		0.21		1.52	

SLIMES-LEACHING AND CEMENTATION DIVISION

The slimes-leaching division (Fig. 8) treats the slimes produced in washing the oxide ore undersize in the oxide sand-washing plant. The slimes-leaching plant was constructed after experience had demonstrated the necessity of removing fines from the regular leaching circuit. Equipment installed for leaching and washing slimes is as follows: two Dorr traction thickeners, 150 ft. in diameter; three slurry mixers, 36 ft. in diameter and 20 ft. deep, and three leaching agitators, 24 ft. in diameter and 16 ft. deep. These, together with the necessary pumping and auxiliary equipment, deliver the leached slime to two rows (three each) of counter-current wash thickeners, 150 ft. in diameter, the two rows

operating in parallel. The wash thickeners are on a gentle slope, making possible a partial gravity flow from the upper to the lower tanks.

Suitable pump houses equipped with pumps for handling overflows and spigot discharges are between the pairs of parallel tanks and are numbered 1, 2 and 3, proceeding downward. The wash thickeners are numbered in the same manner, No. 1 being the head of the washing circuit, from which strong solution is advanced to the precipitation launders.

Equipment ahead of the leaching agitators is not acidproof. The neutral thickeners and slurry tanks are of concrete construction. The moving truss of the thickeners, the agitator mechanism of the slurry tanks, the Wilfley sand pumps and pipe lines are all of iron and steel. Starting at the leaching agitators, all equipment is acidproof. The

Fig. 8.—Slimes-leaching plant and cementation launders.

agitator tanks are concrete, lined with brick laid in mastic mortar, and are equipped with wooden stirring mechanisms and rubber-covered cast-iron rakes. The six 150-ft. thickeners in the washing circuit are of concrete construction, lined with brick and mastic. The portions of the moving trusses that extend into the solution are made of wood. Rubber-covered steel rakes are used. All pumps, pipe lines and valves are of lead. Launders for pulp are made of wood and lined with brick bound with mastic mortar.

The slimes pulp from the sand-washing plant, which contains approximately 10 per cent solids, is delivered by launder to the two neutral thickeners. With tonnages handled to date only one of these thickeners has been required. The clear overflow is piped so that it can be used for make-up in washing the leached slime, wasted, or delivered to the pumping station, which returns it to the general plant-water circuit. The spigot product from the thickeners, which contains approximately 42 per cent solids, flows by gravity to a 4-in. Wilfley sand pump, which elevates it to a distributor box adjoining and above the slurry mixers.

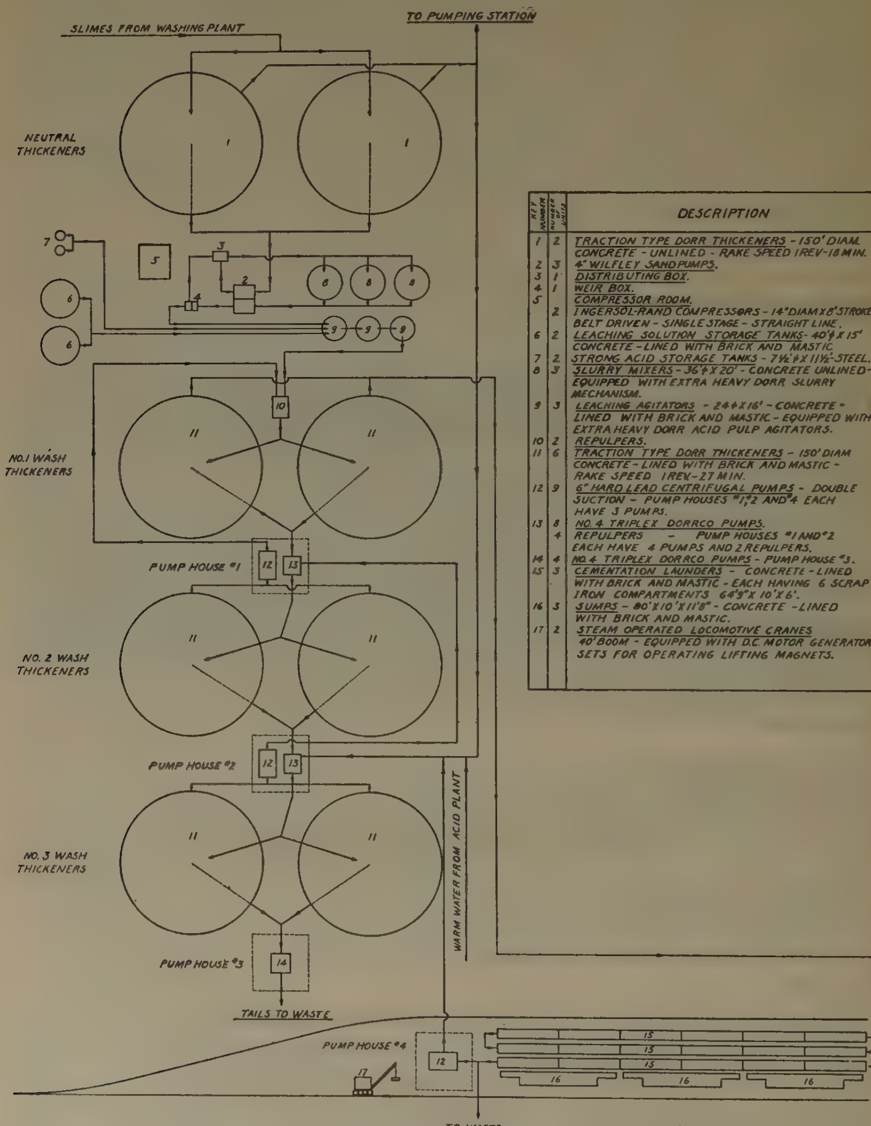
From this box the pulp can be run into any one of the three slurry mixers that serve as storage tanks.

The pulp feed to the leaching agitators is drawn off from the slurry storage tanks over weirs. Weir levels are maintained automatically by motor-controlled pinch clamps. This pulp flows to a 4-in. Wilfley sand pump and is pumped to the No. 1 leaching agitator. The leaching solution and the new acid are added to the feed as it enters the agitators. The leaching solution is made up of high-acid discards from the leaching and electrolytic plants. The three agitators operate in series, and the pulp is advanced by air-lifts and launders, which form part of the agitating mechanism.

The procedure from this point on is merely a counter-current washing operation (Fig. 9). The six 150-ft. thickeners used for washing are arranged in two rows of three tanks each. Each row operates in series and the two rows in parallel. The No. 3 leaching agitator outflow enters the No. 1 repulpers along with the overflow from the No. 2 thickeners. The product from these repulpers is laundered, split and forms the feed to the two No. 1 thickeners. The No. 1 thickener overflows are laundered to the cementation plant. The spigot underflows from the No. 1 thickeners are elevated by 4-in. triplex Dorrco pumps, in No. 1 pump house, and flow into the No. 2 repulpers together with the overflow from No. 3 thickeners. These repulpers furnish the feed for the No. 2 thickeners. The overflows from the No. 2 thickeners are advanced into a tank in No. 1 pump house, which feeds a 6-in. centrifugal lead pump of Anaconda design, and delivered to the No. 1 repulpers. The No. 2 thickener spigot outflows are elevated by 4-in. triplex Dorrco pumps, in No. 2 pump house, and discharged into the No. 3 repulpers with stripped solution from the cementation launders and make-up water. The make-up water comes from various sources and it is advantageous to use the warmest water available. The product from this repulper forms the feed to the No. 3 thickeners. The overflow from these thickeners goes to a 6-in. centrifugal lead pump, in No. 2 pump house, and is delivered to the No. 2 repulpers. The No. 3 thickener spigot discharges are elevated by 4-in. Dorrco pumps, in No. 3 pump house, sampled and run to waste.

The slimes-leaching plant was started up in midwinter and trouble was encountered immediately with poor settling rates in the wash thickeners. This was found to be due largely to the low temperature of the pulp. Steps were taken immediately to remedy this condition. Warm water from the smelter power-house condensers was piped to the washing plant, and warm water from the acid plant to the No. 3 repulpers for make-up water. In addition to raising the temperature of the pulp circuit, the speed of the moving trusses in the wash thickeners was reduced from one revolution in 18 min. to one revolution in 27 min.

Lute wells, installed in the wash thickeners, also materially improved operation. These changes, with better control of the Dorrco pumps, permitted the treatment of all current slimes from the washing plant.



KEY NUMBER	NUMBER OF UNITS	DESCRIPTION
1	2	TRACTION TYPE DORR THICKENERS - 150' DIAM. CONCRETE - UNLINED - RAKE SPEED 1 REV-18 MIN.
2	3	4" WILFLEY SANDPUMPS.
3	1	DISTRIBUTING BOX.
4	1	WEIR BOX.
5		COMPRESSOR ROOM.
	2	INGERSOL-RAND COMPRESSORS - 14" DIAM X 8" STROKE BELT DRIVEN - SINGLE STAGE - STRAIGHT LINE.
6	2	LEACHING SOLUTION STORAGE TANKS - 40' φ X 15' CONCRETE - LINED WITH BRICK AND MASTIC
7	2	STRONG ACID STORAGE TANKS - 7½' φ X 11½' - STEEL.
8	3	SLURRY MIXERS - 36' φ X 20' - CONCRETE UNLINED - EQUIPPED WITH EXTRA HEAVY DORR SLURRY MECHANISM.
9	3	LEACHING AGITATORS - 24 φ X 16' - CONCRETE - LINED WITH BRICK AND MASTIC - EQUIPPED WITH EXTRA HEAVY DORR ACID PULP AGITATORS.
10	2	REPULPERS.
11	6	TRACTION TYPE DORR THICKENERS - 150' DIAM CONCRETE - LINED WITH BRICK AND MASTIC - RAKE SPEED 1 REV-27 MIN.
12	9	6" HARD LEAD CENTRIFUGAL PUMPS - DOUBLE SUCTION - PUMP HOUSES #1, #2 AND #4 EACH HAVE 3 PUMPS.
13	8	NO. 4 TRIPLEX DORRCO PUMPS.
	4	REPULPERS - PUMP HOUSES #1 AND #2 EACH HAVE 4 PUMPS AND 2 REPULPERS.
14	4	NO. 4 TRIPLEX DORRCO PUMPS - PUMP HOUSE #3.
15	3	CEMENTATION LAUNDERS - CONCRETE - LINED WITH BRICK AND MASTIC - EACH HAVING 6 SCRAP IRON COMPARTMENTS 64'9" X 10' X 6'.
16	3	SUMPS - 80' X 10' X 11'8" - CONCRETE - LINED WITH BRICK AND MASTIC.
17	2	STEAM OPERATED LOCOMOTIVE CRANES 40' BOOM - EQUIPPED WITH D.C. MOTOR GENERATOR SETS FOR OPERATING LIFTING MAGNETS.

FIG. 9.—FLOW SHEET OF SLIMES-LEACHING PLANT.

The overflow from the No. 1 wash thickeners flows through a lead-lined launder to the cementation division. This plant consists of three parallel concrete launders which are lined with brick laid in mastic.

They are arranged for a series flow of solution. The upper launder is 3 ft. 3 in. above the middle one, which in turn is the same elevation above the lower launder. There is a 3-ft. space between the launders. Each launder is 395 ft. 6 in. long and is divided into six scrap-iron compartments, the inside dimensions of each being: length, 64 ft.; width, 10 ft. and depth, 6 ft. Along the top and upper side of each cementation terrace there is a small launder for advancing the solution from compartment to compartment, or for bypassing solution when a compartment is being cleaned. The scrap iron rests on false bottoms of 6 by 8-in. timbers spaced 3 in. apart. Underneath the false bottom there are outlets for draining and washing when a compartment is cut out for cleaning. The permanent bottom has a slope of 2½ in. to 1 ft. toward the discharge openings.

Two steam-operated locomotive cranes with 40-ft. booms—equipped with d.c. motor-generator sets for operating lifting magnets—are provided for handling scrap iron and cement copper. Most of the cement copper is hand-shoveled into dump buckets and loaded into railroad cars with the cranes. When the compartments are drained some cement copper escapes with the solution and is caught in settlers on the lower side of the launders. These are excavated periodically with a clamshell bucket.

In passing through the launders, approximately 97 per cent of the copper is precipitated from the solution. All of the stripped solution flows to the No. 4 pump house, which is just below the cementation plant, and most of it is returned to the No. 3 repulpers, in the washing plant, by 6-in. centrifugal lead pumps. Thus the cementation operation is virtually in a closed circuit with the slimes-washing tanks.

Originally the cement copper was mixed with concentrate and smelted in the reverberatory furnace. This practice was abandoned because it resulted in slags of a high copper content. The cement copper is now shipped to the sulfide roaster plant and calcined. The calcined product is leached in agitating tanks with spent electrolyte from the tank house. The enriched solution is added to the purified strong solution as it enters the dechloridizing plant.

Metallurgical data pertaining to the leaching, washing and cementation operations are given in Table 4.

Purification Plant

The purification plant is on the intermediate level between the leaching vats and electrolytic plant. Its function is the preparation of enriched solution from the leaching process for electrolytic deposition of copper through the removal of ferric iron and other deleterious impurities dissolved from the ore.

The purification process consists of the treatment of the so-called strong solution from the vats with finely ground limerock, together with

TABLE 4.—*Slimes Treatment Metallurgical Data*

	July–Dec. 1930	Jan.–June 1931	July–Dec. 1931
LEACHING			
Feed to agitators:			
Dry, tons	55,602	47,386	48,709
−200-mesh, per cent	75.06	72.92	70.47
Pulp, per cent solids	40.91	41.86	42.61
Total copper, per cent	2.38	2.395	2.338
Sulfide copper, per cent	0.13	0.12	0.126
Pulp temperature, deg. C	14.1	14.0	11.8
Solution to agitators			
Volume, tons	50,765	57,341	59,250
Grams per liter copper	9.68	11.26	6.14
Grams per liter H_2SO_4	42.58	56.54	56.91
Temperature, deg. C	18.9	23.0	23.1
New 60° Bé. acid to agitators, tons	2,092.5	502.3	549.3
Pulp from agitators, temp., deg. C	17.7	19.4	18.3
WASHING			
No. 1 thickeners			
Feed, per cent solids	7.77	5.99	6.79
Feed, temperature, deg. C	16.5	15.1	13.8
Overflow, grams per liter copper	3.75	2.87	2.61
Overflow, grams per liter H_2SO_4	2.76	2.45	2.97
Spigot, per cent solids	32.42	40.77	42.58
No. 2 thickeners			
Feed, per cent solids	7.77	5.99	6.79
Feed, temperature, deg. C	16.2	14.8	13.1
Overflow, grams per liter copper	1.62	0.53	0.52
Overflow, H_2SO_4, grams per liter	1.60	1.31	1.62
Spigot, per cent solids	28.07	38.39	36.85
No. 3 thickeners			
Feed, per cent solids	7.77	5.99	6.79
Feed, temperature deg. C	17.7	15.8	14.0
Overflow, grams per liter copper	0.65	0.15	0.14
Overflow, grams per liter H_2SO_4	0.94	1.31	1.55
Spigot (tailing), per cent solids	25.25	34.21	33.99
Tailing			
Total copper, per cent	0.367	0.189	0.225
Acid-soluble copper, per cent	0.155	0.094	0.114
Water-soluble copper, per cent	0.150	0.046	0.047
Sulfide copper, per cent	0.062	0.049	0.064
RECOVERY			
Feed and tailing			
Total copper, per cent	86.27	92.36	90.59
Leach copper, per cent	87.97	94.05	92.86
Over-all, based on total copper in slimes and solutions to plant minus copper in tailing, per cent	90.32	95.68	93.63
ACID CONSUMPTION			
Acid per ton slime, lb. 60° Bé	174.9	199.4	204.7
CEMENTATION DATA			
Solution inflow			
Volume, tons	507,230	673,948	641,184
Grams per liter copper	3.81	2.82	2.48
Grams per liter H_2SO_4	4.22	2.45	2.74
Grams per liter iron	3.85	7.81	8.93
Temperature, deg. C	15.4	14.5	13.1
Outflow			
Grams per liter copper	0.05	0.065	0.069
Grams per liter H_2SO_4	2.26	1.45	1.62
Grams per liter iron	8.49	11.16	11.52
Temperature, deg. C	16.9	15.7	14.3
Cement, per cent copper		72.74	71.52
Copper precipitated, per cent	98.69	97.71	97.20
Scrap-iron consumption, lb. per lb. copper produced	1.353	1.408	1.375

the basic copper carbonate precipitate produced from the ground lime-rock stripping of wash waters from the leaching plant and the purification plant, an operation carried on in conjunction with purification. After the precipitation of impurities from heated strong solution, the solution is filtered through Moore filters and delivered to the dechloridizing plant

and the washed precipitate is discharged to waste. The process includes the following named operations: (1) limerock grinding, (2) stripping, (3) purification, (4) filtration.

The equipment of this plant, except the heating tanks and Dorr thickeners, is contained in a structural steel building. It consists of a system of heating tanks, agitator tanks, Moore filter equipment, Dorr settling tanks, Oliver filters, ball mills for grinding limerock, compressors, vacuum pumps and equipment for handling and distribution of materials and products.

Limerock Grinding

Experience has demonstrated that the limerock used in the purification process should not contain over 20 per cent plus 200-mesh material. If it is coarser, the chemical reactions involved take place more slowly and there is a tendency for the larger particles to settle out of the precipitate.

Three parallel sections are provided for grinding limerock, each containing a 177-ton storage bin, a limerock feeder, a Hardinge mill, 3 by 8 ft., operating in closed circuit with a Hardinge superfine air classifier, and auxiliary equipment.

The crushed limerock (−3/8 in.) is transferred in Ingoldsby ore cars from the smelter flux bin to a track bin of 40 tons capacity, which is above the limerock grinding plant. From this bin it is delivered by an 18-in. belt conveyor to the three storage bins serving the Hardinge mills.

Dry grinding is used and the limerock is fed continuously from the storage bin to the Hardinge mill. The finely ground limerock in the mill is exhausted by a fan, which delivers to the superfine air classifier. The classifier oversize is returned to the mill and the finished product is delivered by a screw conveyor system to storage bins in the stripping and purification divisions. The capacity of each grinding section is about 3½ tons per hour. The ball consumption averages 0.90 lb. per ton of ground limerock.

TABLE 5.—*Analyses of Limerock*

Screen Analysis				Partial Analysis	
Coarse Limerock		Ground Limerock			
Mesh	Cum. Per Cent	Mesh	Cum. Per Cent	Constituent	Per Cent
+0.525 in.	0.37	+100	3.13	CaO	51.7
+ 3 mesh	43.08	+150	8.17	SiO_2	7.0
+ 48 mesh	91.10	+200	17.49	Fe	0.6
− 48 mesh	8.90	−200	82.51	Al_2O_3	0.8

Stripping

The stripping operation recovers the copper from discard and excess wash solutions in a product from which the copper is extracted in the purification operation and made available for electrolytic deposition. The filtered cake from stripping is an efficient precipitant for the impurities in the strong solution; in fact, it is much more desirable than raw limerock.

The solutions treated in the stripping operation consist of excess wash from the Moore filters and discard wash solution from the leaching plant. The solutions are heated to about 37° C. and treated in a series of agitators with an excess of ground limerock. The pulp from the agitators flows to Dorr thickeners. The overflow from the thickeners runs to waste. The spigot product is filtered and used as the precipitant in the purification operation.

The equipment used consists of the following: (1) a concrete tank, 58 ft. in diameter by 20 ft. deep, lined with brick and mastic, used for storage of leaching-plant discard wash water; (2) a concrete tank for storage of Moore-filter excess wash, having a capacity of 30 tons; (3) two 6-in. horizontal centrifugal lead pumps used for delivering solution to heating tank; (4) a lead-lined heating tank, 8 by 10 ft. by 12 ft. deep; (5) eight circular concrete stripping agitators, 10 ft. in diameter by 15 ft. deep; (6) three rectangular concrete stripping agitators, 10 by 10 ft. by 12 ft. deep; (7) two Dorr thickeners, 30 ft. in diameter, tanks and mechanism being of steel; (8) two 3-in. cast-iron Wilfley sand pumps for delivering Dorr spigot discharge to Oliver filters; (9) two Oliver filters, 8 ft. diameter by 12 ft. long, covered with No. 27 cotton cloth; (10) two 20-ton pulverized limerock storage bins equipped with poidometer feeders.

The stripping operation is continuous. The composite of the solutions from the two storage tanks is pumped to the heating tank at the rate of 60 to 80 tons per hour. An average analysis of the solution stripped is as follows: Cu, 8.02 grams per liter; H_2SO_4, 0.39; Fe, 0.46; Al, 0.74. From the heating tank this solution is laundered to the first stripping agitator. The 11 agitators are connected in series and are terraced for gravity flow of solution. Compressed air at 25-lb. pressure is used for agitation. Ground limerock is fed from the storage bins by poidometers into agitators 1, 3, 6, 7 and 8 in amounts required for complete precipitation of the copper. An average of 48 grams of limerock is added per liter of solution treated. The pulp from the last agitator is laundered to two Dorr thickeners operating in parallel, where the overflow is discarded, and the spigot discharge delivered by a 3-in. Wilfley sand pump to a distributing box above the Oliver filters. From this box most of the pulp goes to the Oliver filters but if there is more than is required for the purification operation it is returned to No. 3 stripping agitator.

This pulp filters readily. The filtered cake drops into a launder and is sluiced into the first purification agitator with strong solution. It consists largely of basic copper carbonate, calcium sulfate and excess limerock, and is designated as the "Oliver cake." An average partial analysis of this cake is as follows: 10.28 per cent Cu, 30.16 per cent CaO, 1.10 per cent Fe, 24.84 per cent SO_4.

Purification

In the purification operation the strong solution from the leaching plant is treated in mechanical agitators with Oliver cake and raw limerock. Most of the ferric iron and high percentages of the other impurities that are deleterious to the electrolytic deposition of copper are precipitated. The basic copper carbonate in the Oliver cake is dissolved and thus this copper becomes available for electrolytic deposition. The pulp containing the precipitated impurities goes to the Moore filter plant.

The purification circuit consists of a heating tank, agitating tanks and a surge tank for purified pulp. The heating tank is of wood and is lined with lead. It is 8 ft. 3 in. by 20 ft. by 8 ft. deep and contains 20 lead steam coils having a total heating surface of 687 sq. ft. Steam at 18 to 20 lb. pressure is supplied by two 500-hp. Badenhausen boilers, which are in a building adjacent to the purification plant. There are 10 concrete agitator tanks, connected in series and arranged for a gravity flow of solution. The two tanks at the head of the circuit are rectangular with a common dividing wall. Each is 22 ft. 6 in. long by 8 ft. 4 in. wide by 19 ft. 6 in. deep; each has a capacity of 110 tons and is provided with two mechanical agitators. The remaining eight tanks are circular and are arranged in two parallel rows of four each. Each one is 10 ft. in diameter and 15 ft. deep, has a capacity of 35 volumetric tons, and is equipped with a single chrome-iron agitator that has a peripheral speed of 1000 ft. per minute.

Two steel bins equipped with poidometers are provided for feeding ground limerock; one at the head of the circuit and the other over the circular agitators. Five rectangular tanks, having a total capacity of approximately 200 tons of solution, were provided for aging the pulp before filtration but experience has demonstrated that this conditioning period does not improve the filtration rates, therefore it has been discontinued.

The concrete surge tank for the purified pulp ahead of the filters is 29 ft. 2 in. by 9 ft. 6 in. by 18 ft. 4 in. deep. An 8-in. lead manifold, extending the entire length of the filter floor with connections to each loading tank, delivers the pulp from the surge tanks to the filters.

The purification operation is continuous. The strong solution flows by gravity from the leaching-plant storage sumps through an 8-in.

lead line to the heating tank at the head of the purification circuit, where it is heated to about 39° C. The normal rate of flow is 150 tons per hour.

Most of the solution from the heating tank flows through a weir box into the first purification agitator. The filtered cake from the stripping operation is added here. When the supply of Oliver cake is insufficient for purification requirements, raw ground limerock is also introduced at this point. The reactions, which start immediately, result in the solution of the basic copper carbonate from the Oliver cake and the precipitation of most of the ferric iron, arsenic, phosphorous, molybdenum and about 20 per cent of the aluminum. The pulp leaving the second agitator is usually in a more or less overpurified condition; that is, an excess of the precipitant has been added and some of the copper precipitated with the impurities. This condition is corrected by adding fresh strong solution to one or more of the circular agitators in sufficient quantity to redissolve the copper and consume the excess limerock. Provisions are made for drawing fresh strong solution from the heater tank to any one of the eight circular agitator tanks. Raw limerock is added at times to the lower end of the purification circuit to correct an underpurified condition, resulting from faulty control at the head end of the circuit. In general, it has been found that the addition of raw limerock to the lower agitators in sufficient amounts to precipitate iron to any extent tends to precipitate copper, and also slows up filtration rates.

With a flow rate of 150 tons per hour the solution remains in the purification agitators approximately 3 hr. The pulp from the last agitator flows through a 10-in. lead pipe to the surge tank that feeds the Moore filters.

A close control of the purification operation must be maintained at all times. For controlling the degree of purification, the iron content of the filtered purified solution is determined every half hour. The aim is

TABLE 6.—*Purification Data*

	Solution Analysis, Grams per Liter		Impurity Elimination, Per Cent
	Unpurified Strong Solution	Filtered Purified Solution	
Cu	41.55	42.09	
H_2SO_4	14.11	0.00	
Total Fe	7.03	1.50	82.12
Ferric Fe	6.32	0.94	87.98
Al	4.99	4.13	21.85
As	1.40	0.11	92.32
P_2O_5	1.52	0.20	87.14
Cl	0.29	0.46	
Mo	0.033	0.003	91.12

to hold the total iron content of the purified solution at from 0.10 to 0.20 grams per liter above the known ferrous iron content of the raw strong solution. A colorimetric copper assay is run hourly on the washed precipitate. If an overpurified condition exists it is shown by this determination. This is important, as any copper leaving the purification circuit in a precipitated condition is lost in the Moore filter cake. Filtration rates on the purified pulp are determined hourly on laboratory Buchner filters. These aid the operator in adjusting the solution flow rate, and in operation of the Moore filter. Limerock consumption, practically all in the form of Oliver cake, averages about 40 grams per liter of strong solution treated. Average purification data are shown in Table 6.

FIG. 10.—MOORE FILTER SECTION, PURIFICATION PLANT.

Filtration

The solution that contains the precipitated impurities is filtered on Moore filters. The filtered strong solution is pumped to a storage tank at the dechloridizing plant; the excess wash solution is transferred to the stripping division and the filter cake runs to waste.

The Moore filter division (Fig. 10) occupies the lower level of the purification building and covers a space 42 ft. 6 in. wide by 252 ft. long. A crane runway, with a span of 42 ft., extends the full length of this section. Three two-motor overhead traveling cranes, each equipped with four 10-ton hooks, transfer the filter baskets.

There are five filter sections, each made up of three adjoining concrete tanks and two Moore filter baskets. One tank is for loading the pre-

cipitate, the second for washing the cake and the third for final wash and blowoff. Three lead-lined concrete tanks between the filter sections are used for conditioning the filters. The 18 concrete tanks are built in one block. The longitudinal axis of the tanks is at right angles to the crane run. The 15 tanks used for loading and washing are 28 ft. 9 in. long by 9 ft. 8 in. wide by 16 ft. maximum depth. Each of these tanks has three hoppers in the bottom which are provided with compressed-air and drain connections. The three lead-lined conditioning tanks are of the same dimensions except that they are 8 ft. deep and have flat bottoms.

There are 11 Moore filter baskets, two per section and one spare. Each basket has 64 leaves spaced at 5-in. centers. Each leaf has an effective filter area of 98 sq. ft., which is equivalent to 6272 sq. ft. per basket.

The individual leaf consists of a U-frame, 8 ft. wide by 7 ft. 9 in. high, made of 1-in. chrome-iron pipe, and 90° chrome-iron elbows. The filter bag, made of No. 5 refinery cloth, is slipped over this frame, sewed around the edges and clamped at the top with chrome-iron straps. The leaves are suspended from the main filter frame by chrome-iron bolts. There is a 5-in. lead manifold on each side of the filter frame, to which the vertical legs of each leaf are connected by 1¼-in. rubber hose.

The two manifolds on the filter frame are joined at both ends by lead headers. At one end of the filter, a 4-in. reinforced rubber vacuum hose, 42 ft. long, is attached to the header. This length of hose permits moving the basket from tank to tank in the section without disconnecting. Each 4-in. vacuum hose is connected to a separate valve nest consisting of four rubber-diaphragm lead valves. One of these valves connects with a 12-in. lead vacuum line, which conveys the purified strong solution to a chrome-iron receiver 5 ft. in diameter by 8 ft. high; the second to an 8-in. lead vacuum line which delivers the wash solution to a chrome iron receiver 4 ft. in diameter by 8 ft. high; and the third and fourth to low-pressure air and water lines which are used for blowing off the cake. In order to obtain a higher filtration rate during the loading period, an 8-in. auxiliary vacuum line is provided at each loading tank to connect the 12-in. vacuum line to the lead header on the filter frame. The tops of the two receivers are connected by chrome-iron pipe to a bronze vapor condenser, 4 ft. in diameter by 8 ft. high; and thence through a moisture trap, 5 ft. in diameter by 9 ft. high, and pipe to the vacuum pumps. Barometric legs of 8-in. lead pipe transfer the solutions from the condensers to separate pump sumps below the Moore filters. The purified strong solution is pumped to a concrete storage tank, 50 ft. in diameter by 10 ft. deep, situated at the north end of the dechloridizing plant, by two 8-in. horizontal centrifugal lead pumps. The wash solution from the filters is pumped by 6-in. centrifugal lead pumps to a concrete tank, 10 by 10 ft. by 12 ft. deep, beside the surge tank that feeds the filters. A portion of this wash water is used again in

washing the Moore cake, and the excess overflows into a launder that delivers it to the 30-ton storage tank in the stripping division.

Vacuum is supplied by three Worthington duplex direct-connected pumps, each 27 by 14 in. Five Ingersoll-Rand, 19 by 14-in. belt-driven duplex compressors are provided, which furnish 30-lb. air to the oxide division of the reduction plant. The air used by the Moore filters is reduced to 5 lb. pressure.

The time allowed for the filter cycle is 80 min. With five sections operating, this means that one of the filters starts on the cycle every 8 min. Each of the 10 baskets operates on a definite and rigid schedule. The cycle is divided as shown in Table 7.

TABLE 7.—*Filter Cycle*

	Minutes
In loading tank	22
Transfer from loading to wash tank	2
In wash tank: advance wash	4
regular wash	17
Transfer from wash to blowoff tank	2
In blowoff tank: final wash	22
blowoff water	2
blowoff air	2
Transfer to loading tank and emergencies	7
Total	80

The purified pulp flows through the lead manifold from the surge tank of the purification division to the loading tanks as needed. The pulp in the filter tanks is agitated continuously with air. As soon as the basket is placed in the loading tank, vacuum is applied to the leaves through the regular 4-in. hose and the 8-in. auxiliary connection, both of which are open to the 12-in. strong-solution vacuum line. The vacuum at the receivers averages 18 to 20 in. of mercury. During the loading period the vacuum on the filter manifolds varies from 3 in. at the start to 16 in. at the finish. At the end of the loading period the valve on the auxiliary vacuum line is closed and the basket transferred by the crane to the washing tank. This tank usually is supplied with filter wash solution, but fresh water can be added if necessary. With the vacuum still open to the strong-solution system, a 4-min. period of advance wash is taken in order to displace the high-copper solution remaining in the cake and filter leaves. The vacuum is then changed to the wash solution system, and wash water pulled through the cake for 17 min. The basket is then transferred to the blowoff tank, to which fresh water at 25° to 30° C. is added. The washing continues for 22 min. in this tank, then the vacuum valve is closed and the cake blown off. In this operation water at about 7 lb. pressure is applied internally to the leaves through the 4-in. vacuum hose for a short period. The water

lubricates and loosens the cake. This is followed by air at about 5 lb. pressure. The blowoff requires from 2 to 4 min. The basket is then returned to the loading tank.

The cake that accumulates in the blowoff tanks is discharged periodically. It is sampled by an automatic sampler and runs to waste. Approximately 58 per cent of the cake is $CaSO_4$, having an average analysis as follows: total copper, 0.86 per cent; water-soluble copper, 0.43; iron, 7.77; aluminum, 1.30; SO_4, 44.39; CaO, 23.44.

The copper loss in the cake amounts to approximately 1.7 per cent of the copper entering the purification plant. The character of the precipitate largely determines the washing efficiency. Slimy precipitate obtained from solutions of high iron content is very difficult to filter, wash and blow off. The washing period is held at the maximum at all times.

The average rate of filtration is 5.31 liters of purified solution per square foot of filter area per cycle. The thickness of the cake is about ⅜ in. Approximately 86.73 grams of cake is obtained per liter of purified solution, or 1.02 lb. per sq. ft. of filter area per cycle.

The filtration bags gradually become hard and the filtration rates diminish; also, the hardened bags break and tear easily as the cotton fabric becomes impregnated with insoluble iron compounds and calcium sulfate. The higher the temperature of the pulp to filtration, the faster this action takes place. As the filtration rate is best at high temperatures, it is necessary to compromise between hardened bags and slow filtering rates. The pulp is filtered at about 36° C. At this temperature the filtration rate is fair and the bags do not harden very rapidly.

The filters are conditioned daily in a solution containing 50 grams of sulfuric acid per liter. This is a cheap and fairly effective scavenger for removing slime that adheres to the fabric but it will not remove the impregnated salts to any extent.

Fig. 11 is a diagrammatic flow sheet covering both the stripping and purification sections of the purification plant.

Dechloridizing Plant

The purified strong solution from the purification plant contains approximately 0.46 grams of chlorine per liter. In order to avoid the deposition of cuprous chloride on the cathodes in the tank house, and the subsequent volatilization loss during furnace refining, it is necessary to reduce the chlorine content of the solution to about 0.20 grams per liter prior to electrolysis. This is done in the dechloridizing plant, where the chlorine is precipitated as cuprous chloride. The solution is agitated with an excess of cement copper and the resulting pulp allowed to settle. The clarified solution flows by way of storage tanks to the electrolytic plant. The settled residue, which consists of cuprous chloride and cement copper, is treated with a ferrous chloride solution, which dissolves the

cuprous chloride. The copper in this solution is precipitated on scrap iron, washed and re-used for dechloridizing.

The dechloridizing plant is about 125 ft. south of the purification plant, at an elevation that permits a gravity flow of solution to the tank

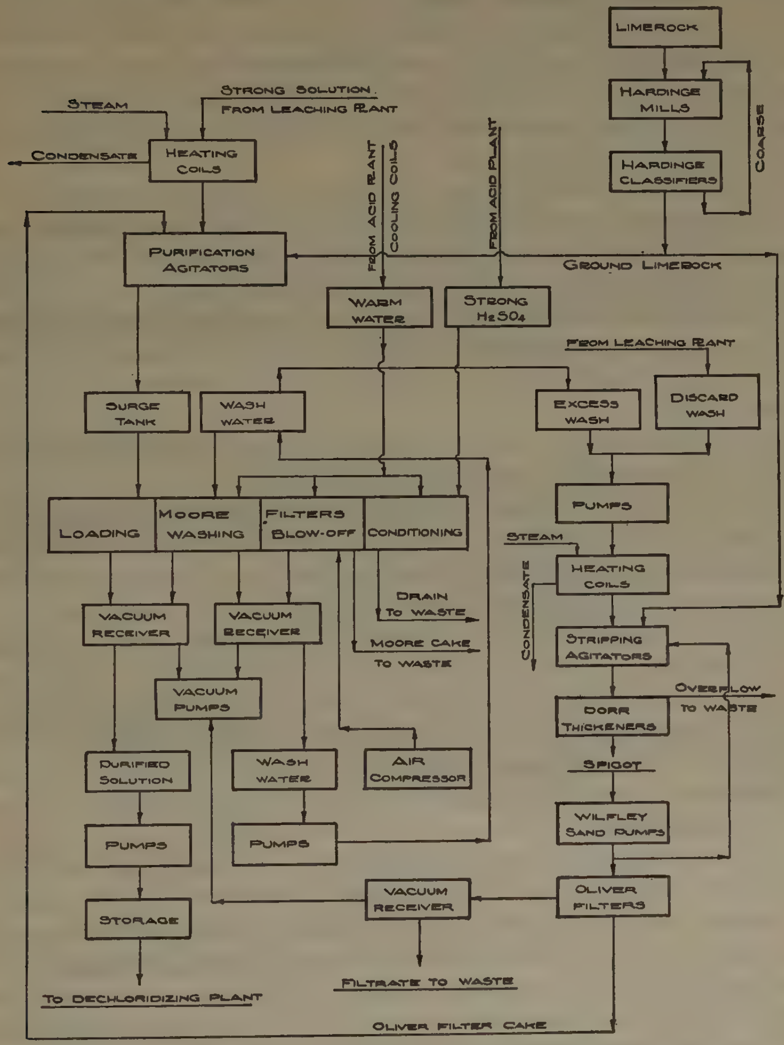


Fig. 11.—Flow sheet of purification plant.

house. The equipment in this plant includes: (1) three brick-lined concrete emulsifier tanks, 6 ft. 3 in. wide by 7 ft. long by 6 ft. 6 in. deep, each provided with a bronze mechanical agitator, one emulsifier being used for dechloridizing, the second for dissolving cuprous chloride and the third for washing cement copper; (2) two brick-lined concrete agitator tanks used for dechloridizing, 10-ft. dia. by 15 ft. deep, each equipped with

chrome-iron agitators; (3) three brick-lined concrete tanks, 39 ft. 6 in. long by 33 ft. wide by 5 ft. 4 in. deep, used for the settling of cuprous chloride; (4) four brick-lined concrete settling tanks, 37 ft. 8 in. long by 11 ft. 6 in. wide by 6 ft. deep, two being used for unwashed cement copper and two for washed cement copper; (5) two bronze cementation drums, 7-ft. dia. by 10 ft. long, lined with cast copper sectors, which are charged with scrap iron and revolve at a speed of 8 r.p.m.; (6) four 2-in. rubber-lined pumps for handling ferrous chloride solution; and (7) two brick and mastic-lined concrete tanks, 50-ft. dia. by 10 ft. deep, used for the storage of dechloridized strong solution.

The emulsifiers, agitators and cementation drums are contained in a steel building 32 ft. wide by 42 ft. long. The row of 10 adjoining settling tanks on the lower side of the building is spanned by an overhead traveling crane equipped with a 10-cu. ft. bronze clamshell bucket for excavating cuprous chloride and cement copper. The pumps are housed in a small building below the settling tanks. The two storage tanks for the dechloridized strong solution are just above the tank house and feed the electrolytic tanks.

The purified strong solution flows by gravity through an 8-in. lead line from the concrete storage tank to a weir box in the dechloridizing plant (Fig. 12). The weir overflow, which averages approximately 150 tons per hour, is split. About 5 per cent of it flows continuously to one of the emulsifiers to which washed cement copper is added periodically by the crane. The emulsifier outflow unites with the remainder of the weir discharge and flows into the first agitator tank. As the solution enters this agitator enough sulfuric acid is added to raise the acid content to 10 to 15 grams per liter. Most of the dechloridizing action takes place in the two agitators that operate in series. The overflow from the second agitator is laundered to the cuprous chloride settlers. The clarified overflow, which contains about 0.20 grams chlorine per liter, flows to the two dechloridized strong-solution storage tanks.

Periodically each of the cuprous chloride settlers is cut out of the circuit and cleaned. The settled residue is transferred by the crane to the second emulsifier, and treated with ferrous chloride solution. The cuprous chloride is dissolved and the resulting solution flows into the cementation drums, where the copper is precipitated on scrap iron. The drums discharge into short launders that convey the pulp to the settling tanks for unwashed cement copper. The settler overflow contains a small amount of copper and about 50 grams of chlorine per liter. It is returned to the emulsifier by the rubber-lined pumps and used again for dissolving cuprous chloride.

The settled cement copper from the drums is washed with water in the third emulsifier and settled in the two settling tanks for washed cement copper. The overflow from these settlers, with discard ferrous

chloride solution, passes through a scrap-iron cementation launder before running to waste. The washed cement copper, after settlement, is used again for dechloridizing. This wash removes most of the chlorides and soluble iron salts.

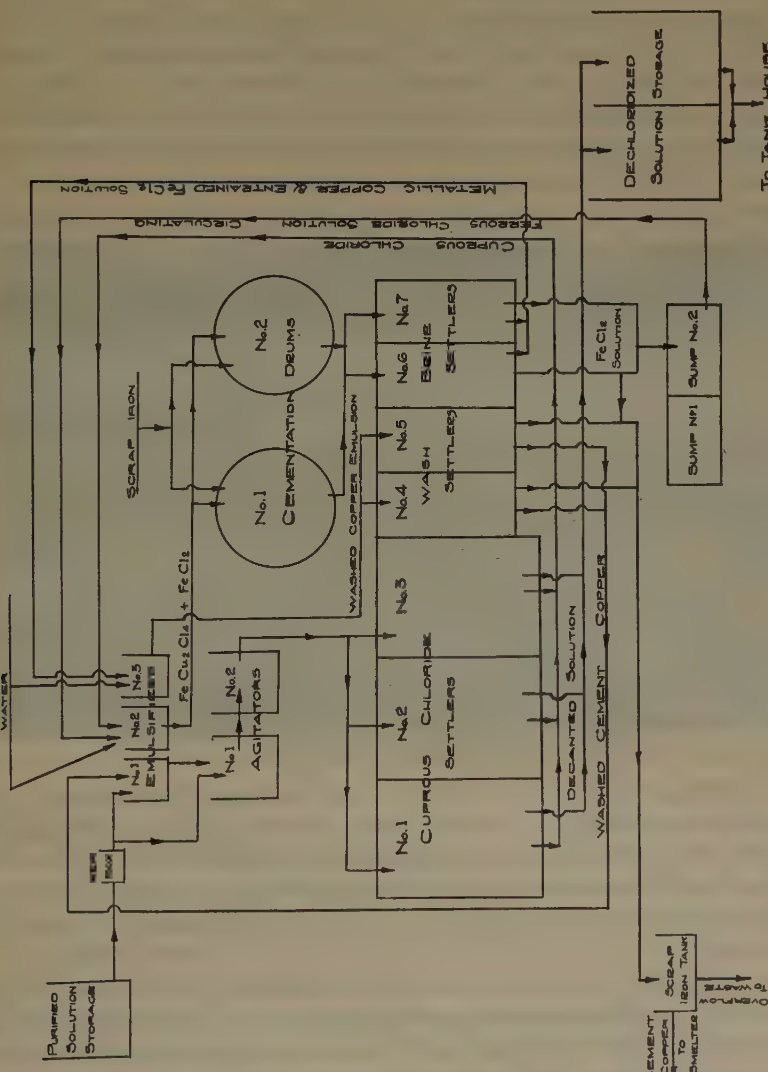


FIG. 12.—FLOW SHEET OF DECHLORIDIZING PLANT.

ELECTROLYTIC PLANT

The electrolytic plant was designed for a daily deposition of 100 tons of copper from leaching-plant solutions, and approximately 7 tons of starting sheets from blister-copper anodes. It is enclosed in a steel

building, 192 by 302 ft., which has a basement and an operating floor. The walls are sheathed with corrugated iron and roofed with corrugated metal protected by asbestos. Ample window area gives inside lighting, and monitors provide ventilation. Foundations and tank-supporting piers are of reinforced concrete.

The electrolytic tank room is on the main floor. Lengthwise it is divided into three crane bays; each bay is 60 ft. 9 in. wide and is served by a 10-ton Shaw stiff-leg overhead direct-current crane. Running the width of the building at the two ends and center are the work floors where starting sheets, anodes and cathodes are handled. These floors are of heavy wood construction with mastic covering and are supported on concrete piers.

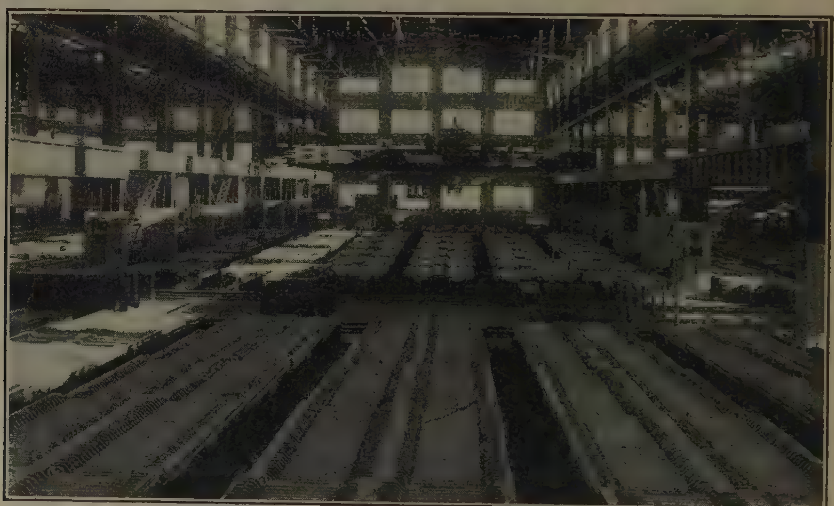


FIG. 13.—CENTER CRANE BAY OF TANK HOUSE.

The pumping equipment, air-lifts, receiving tanks and solution lines, as well as main electrical busbars, are in the basement. The basement floor is brick, laid in mastic. It slopes to launders in the center of the building which drain to two sumps from which solution and wash water are pumped, by means of two vertical lead pumps, to waste or back into the circuit.

Adjoining the electrolytic plant is the substation, a steel building 40 by 141 ft., in which are four motor-generator sets. Each set comprises a motor driving two generators on the same horizontal shaft. Each d.c. generator will deliver 3750 amp. at 425 volts. There are three electrical circuits in the tank house. One motor-generator set is provided for each of the three commercial circuits, and to either of two of these the starting-sheet division is connected in series. The fourth motor-generator set is a spare machine that can be used for any circuit. At normal load these machines operate with an efficiency of approximately 92 per cent.

The busbars that enter the tank-house basement from the substation are suspended from insulated hangers. The center point of each electrical circuit is grounded through a low resistance in order to protect the workmen. Each of the three electrical circuits corresponds to a solution circuit for the commercial deposition of copper. The starting-sheet division, with a closed solution circuit, operates electrically in series with one of the commercial circuits.

Commercial Division

There are 576 electrolytic tanks in the commercial or insoluble-anode division, of reinforced concrete lined with 8-lb. 6 per cent antimonial lead. They rest on lead-capped concrete piers and have a porcelain insulating block under each corner. Inside dimensions are: length, 10 ft. 3 in.; width, 2 ft. 10 in.; depth, 3 ft. 9 in. Side walls are 5 in. thick and the ends and bottoms 4 in. thick. Each tank contains 34 lead anodes, spaced at 3½-in. centers, and 33 cathodes. The insoluble lead anodes were cast from International antimonial pig lead and have the following average composition:

	Original Anodes	Anodes Recast
Lead, per cent	82.12	83.5
Antimony, per cent	15.50	15.0
Arsenic, per cent	1.68	1.3
Copper, per cent	0.44	0.15

The anodes are 39¼ in. long by 26½ in. wide by ½ in. thick, and weigh approximately 200 lb. each. As originally cast, the portion of the supporting bar between the edge of the tank and the anode proper was exposed. In time the spray caused by the evolution of gas at the anodes corroded the bars to such an extent that the bars failed, and some of the anodes dropped into the tanks. This difficulty has been overcome by revising the anode mold so that the lead is now cast around the supporting bar to a point just far enough from the end to permit the insertion of the crane hooks. At present 60.7 per cent of the lead anodes have been recast in this manner and 63 per cent of the anodes have been equipped with porcelain insulators—three complete insulators per anode. These are effective in preventing short circuits and are well worth while. The anodes have shown no tendency to warp. The breakage has been low except for that caused by the corrosion and failure of the supporting bars. Anode corrosion has amounted to 0.844 lb. per ton of copper deposited.

The starting sheets are 28 by 37½ in. They are suspended by two loops, from 7⁄16 by 1 11⁄16-in. rolled copper bars, which are flattened at one end to make good contact. The sheets extend to within 5½ in. of the tank bottom.

The three commercial circuits are equipped and operated in the same manner. Each circuit contains 192 electrolytic tanks. The tanks

are first grouped into sections of 12 tanks each, and then into divisions composed of four or six parallel sections. The tanks in each section are arranged in cascades of six tanks, two cascades to a section. Parallel tanks in a section are spaced 2½ in. apart. There is a drop of 4 in. between tanks, or 20 in. in a cascade of six. Sections within the division are separated by 28-in. walkways, which are approximately 24 in. below the tops of the tanks. Each commercial circuit consists of three divisions, the first two in a circuit containing six sections, and the last one only four sections. The electrolyte flows through the three divisions of the circuit in series and through the cascades of each division in parallel.

A receiving and a distributing tank are at the head of each division. The receiving tank is in the basement and all solution to the division flows into it. The distributing tank is on the main floor and feeds the head tank of each cascade in the division.

The tanks of each commercial circuit are electrically connected in series, and the electrodes in each tank are in multiple. The heavy, cast copper busbars, which rest on insulating boards on top of the outer walls of the tanks, are connected to the main busbars in the basement by copper risers. A rolled copper plate resting on an insulating board supported by the two inner tank walls forms the common contact between tanks. Flat contacts are used.

The dechloridized purified solution, which forms the feed to the tank house, is mixed with acid and a portion of the spent electrolyte, to form the feed solution to the electrolytic tanks. This permits operating at a higher solution temperature and acidity, and results in a lower voltage. It also increases the circulation rate of the electrolyte and aids in the production of a smoother deposit of copper. The flow rate of the electrolyte is about 25 gal. per min. per tank in the first two divisions of a circuit, and 37 gal. per min. in the last division. The amount of acid added to the electrolyte at the tank house is governed by requirements of the leaching plant.

The dechloridized purified solution, which averages approximately 42 grams per liter copper, and 15 grams per liter H_2SO_4, flows from the storage tanks at the dechloridizing plant through an 8-in. lead pipe to weir measuring boxes in the tank house. Three weir boxes are provided, one for each of the head divisions of the three commercial circuits. Sulfuric acid is added to the purified solution at each weir box and the solution then flows into the receiving tank below, where it is mixed with approximately 65 per cent of its volume of spent electrolyte carrying about 11 grams per liter copper and 75 grams per liter H_2SO_4. The resulting mixture, which contains 28 to 30 grams per liter copper, and 45 to 55 grams per liter H_2SO_4, forms the feed to the first division of the circuit.

The receiving tank feeds an air-lift that pumps the mixed electrolyte to the distributing tank from which the electrolyte flows through a lead

manifold to the head tank of each cascade in the first division. The manifold has composition risers to which are attached gum-rubber hoses and pinch clamps for the regulation of the flow to each cascade. The electrolyte flows through the six tanks of each cascade into a rubber-lined wooden launder and from there through a 6-in. lead line to the receiving tank of the second division of the circuit. The electrolyte is advanced through the last two divisions of the circuit in the same manner. Spent electrolyte from the last division goes to a splitting tank, where it is divided, and approximately 40 per cent is returned to the electrolyte circuit and 60 per cent pumped to the leaching plant by two 8-in. horizontal centrifugal lead pumps.

The normal period of deposition is eight days. Cathodes weigh approximately 100 lb. each. In pulling the copper, a unit of four electrolytic tanks is cut out at a time by means of short jumper bars, after which the cathodes are lifted out by the crane, one tank at a time. They are washed in a spray tank and unloaded on to a cathode loading machine, which in turn places them on the cars of the industrial railroad. Each car is weighed and the cathodes again washed before delivery to the casting plant. On its return the crane picks up a load of sheets and places them in the tank from which the cathodes were removed. A partial chemical analysis of the cathodes follows: copper, 99.927 per cent; arsenic, 0.0006; antimony, 0.0003; chlorine, 0.0087.

Originally all cathode inspection was done by hand, but by this method many short circuits escaped the attention of the inspectors. A cathode might be touching the anode slightly but this could not be detected by the inspector. To correct this a system of crane inspection was adopted, which means the lifting of all the cathodes with the crane, half of the cathodes from a tank at a time, straightening the sheets, removing the sprouts and cleaning the contacts. All hand inspection, except when placing sheets, has been discontinued. The schedule entails five complete crane revisions during an eight-day period. The first crane inspection is made 16 hr. after the sheets are placed. Better straightening can be obtained at that time than on the original sheet because it is much stiffer. The second inspection is made 2½ days after the sheets are placed, the third is made on the fourth day, the fourth is made on the fifth day of cathode life, and the fifth 16 hr. prior to pulling the cathodes.

In order to reduce the amount of fine mist thrown off by the gas liberated at the anodes, the electrolyte in all commercial tanks is kept covered with a film of oil. A cheap grade of red engine oil is used and the consumption amounts to 0.42 gal. per ton of copper deposited.

A small laboratory is maintained at the tank house. Hourly samples of feed solution and spent electrolyte are run for copper, and necessary adjustments to flow rates made accordingly. Section, division and over-all voltages are recorded hourly, as well as the total ampere load.

The chemical analysis of tank-house solutions, and power data are given in Table 8.

TABLE 8.—*Tank-house Data*

SOLUTIONS

	Dechloridized Purified Solution	Feed to Electrolytic Tanks	Spent Electrolyte
Copper, grams per liter	42.59	29.2	10.71
H_2SO_4, grams per liter	14.38	47.8	75.95
Total iron, grams per liter	1.73		1.73
Ferric iron, grams per liter	trace		1.08
Chlorine, grams per liter	0.20		
Temperature, Deg. C	27.9	32.5	38.6

POWER

Current density, amperes per sq. ft	10.69
Average tank voltage	1.89
Ampere efficiency, per cent	85.63
Copper deposited: per kw-day d.c. at tanks, lb	27.90
per kw-day d.c. at generator, lb	27.77
per kw-day a.c. at substation A, lb	25.03
Kilowatt-hours a.c. per pound copper deposited	0.958
A.c.-d.c. conversion efficiency, including line loss	90.12
H_2SO_4 regenerated per pound copper deposited, lb	1.53

Starting-sheet Division

The starting-sheet division consists of 4½ sections, or 54 electrolytic tanks, situated at the south end of the center crane bay. Except for small variations these tanks are constructed and arranged in the same way as the commercial tanks. Each tank holds 25 soluble anodes spaced at 4.8-in. centers and 24 starting sheet blanks. The anodes are unrefined blister copper from the smelter. The blanks are ⅛-in. rolled copper sheets with over-all dimensions of 28¾ by 43⅝ in., to which the supporting bars of copper are riveted with joints soldered. Milled grooves near the edge of the blank, on each side and bottom, produce a 28 by 37½-in. starting sheet.

In stripping, one-half of the blanks in a tank are removed at a time by the crane and transferred to a stripping rack. Tanks are not cut out during the operation, density being doubled on the blanks remaining in process. The upper corner of the sheet is loosened by a chisel-like steel tool bent in the form of a hook. Wedge-shaped, short wooden implements are used to complete the removal. With two stripping racks in service, the blanks from all 54 tanks are stripped during one 8-hr. shift. A light engine oil with good filming properties, applied with felt rollers, is most satisfactory for oiling the blanks. The deposit is

dense and firm, and stuck sheets occur rarely. Good sheets produced average about 99.5 per cent. Approximately 11 per cent of the sheets are cut into strips, which are annealed in a small oil-fired furnace, looped and clipped to the starting sheets with a Morrow clip machine. When the electrolyte in the commercial circuits contains 2 grams per liter, or more, of ferric iron, it is necessary to paint the lower part of the loops with an acid-resisting paint.

Under normal conditions a current density of about 18.0 amp. per sq. ft. is maintained. This allows the stripping of 24-hr. sheets weighing approximately 8 lb. each. Of this weight about 3 per cent is trimmings, which are shipped to the casting plant. At this density the voltage drop per tank is about 0.33 volt. Ampere efficiency varies from 85 to 90 per cent and power yield from 160 to 170 lb. per kw-day d.c. at the generator.

An average analysis of the starting sheets shows: copper, 99.935 per cent; iron, 0.0011; chlorine, 0.0007; arsenic, 0.0016; antimony, 0.0006; silver, 0.15 oz. per ton; gold, trace.

The anodes are cast with heavy supporting lugs and weigh from 550 to 600 lb. each. They are the same width as the blanks and extend to the same depth in the tanks. They are rough, porous and vary considerably in thickness. Anode scrap returned to the smelter amounts to approximately 20 per cent of the original weight. There is no fixed anode-loading schedule. They are inspected and changed from day to day as required. A typical partial analysis of the anodes is as follows: copper, 99.30 per cent; sulfur, 0.49; iron, 0.048; arsenic, 0.041; antimony, 0.002; silver, 3 oz. per ton; gold, 0.27 oz. per ton.

Slimes are cleaned from the tanks at intervals of about two months and at such times the division is shut down for two days. During the first 24 hr. the tanks are cleaned and the electrodes rearranged. On the second day loss of temperature and shrinkage of the electrolyte are made up. Slimes removed from the tanks are settled, partly dried and shipped to the smelter. Approximately 0.027 ton slimes is produced per ton of starting sheets stripped.

The starting-sheet solution circuit is closed. Means are provided for the discard of impure electrolyte to the commercial circuit. Make-up consists of dechloridized purified solution, water and acid. The electrolyte is heated by lead steam coils in a 6 by 6 ft. head tank. From this tank it flows through an 8-in. manifold connected with hard-rubber risers to the upper tanks of the nine cascades comprising the division. Passing through the six tanks of each cascade it is discharged through composition fiber pipes to a rubber-lined wooden launder from which it flows by gravity to a 9 by 9-ft. receiving tank in the basement. A 2-in. rubber-lined pump returns it to the head tank. Inflow is at the surface of the electrolytic tanks. The outflows are baffled by lead plates, which

cause the solution to be drawn off to the next tank at a point 5 in. from the bottom of the tank. The temperature of the electrolyte leaving the heating tank is approximately 59° C. In passing through the electrolytic tanks a temperature drop of from 10° to 12° C. takes place. Flow rate is maintained at 10 U.S. gal. per minute per cascade of six tanks.

The only addition reagent used is glue, which is introduced over the 24-hr. period at the rate of 0.05 lb. per tank-day. The glue in solid form is placed in the outflows of the cascades in perforated lead cups and is dissolved slowly by the electrolyte.

The average composition of the electrolyte is as follows: copper, 40.90 grams per liter; H_2SO_4, 150.70; iron, 1.19; chlorine, 0.03; HNO_3, 0.06; arsenic, 4.09; antimony, 0.14.

Furnace-refining and Casting Plant

The plant for fire-refining and casting electrolytic copper into marketable shapes is on the same level and approximately 350 ft. north of the electrolytic plant. A 30-in. gage industrial railway serves the two divisions. Haulage equipment includes 38 cathode cars, 3 anode cars and two 5-ton electric locomotives. The space between the two plants has ample trackage facilities and affords a suitable location for the storage of cathodes.

The furnace refinery is housed in a steel building 135 ft. wide by 187 ft. long. Equipment consists of one reverberatory refining furnace, a waste-heat boiler, a cathode-charging crane, a casting wheel, a bosh conveyor, an inspection conveyor, a mold press and accessories. On the charging side of the furnace there is an aisle 39 ft. wide which is spanned by a cathode-charging crane of the suspended type, of 3½-ton capacity. The casting and inspection aisle, which is on the tapping side of the furnace, is 70 ft. wide and is spanned by a 5-ton crane that serves the casting wheel, mold press and inspection conveyor.

The refining furnace, which has a hearth dimension of 12 by 29 ft., is constructed of silica and magnesite brick. It is held together in the usual manner by heavy buckstays and tie-rods. Cast-iron plates just inside the buckstays extend all around the furnace and form a backing for the furnace bottom and walls to above the metal line. The inverted silica-brick arch that forms the furnace bottom rests on a reinforced concrete foundation, the top portion of which is air-cooled. The bottom is 32 in. thick, and is made up of two courses of silica brick, 20 in. and 12 in., respectively. The furnace walls are 24 in. thick and are built of silica and magnesite brick. Below the metal line, the magnesite-brick lining is 18 in. thick and above it is only 9 in. thick. The furnace roof is made of 15-in. silica brick.

The furnace is fired with fuel oil, one No. 2N Best burner being used. The oil is heated to 140° F. and maintained at 100 lb. pressure; 90-lb.

air is used for oil atomization. A No. 4½ Sirocco blower furnishes secondary air for combustion as well as air for cooling the furnace bottom. The gases from the furnace ordinarily pass through a 400-hp. Stirling waste-heat boiler of 400 lb. working pressure, to the stack; but a bypass flue to the stack is provided. The firebrick-lined steel stack is 5 ft. inside diameter and 170 ft. high.

The copper is tapped from the furnace into pouring ladles that serve a 40-ft. diameter Clark casting wheel. This wheel, which is equipped with 30 wire-bar molds, discharges on to the bosh conveyor, which elevates and delivers the bars to a 50-ft. inspection conveyor. A pneumatic hoist at the end of the inspection conveyor loads the good bars on to 3-ton Baker trucks. This copper is then weighed on two platform scales (that is, double weighed) and stacked on the copper dock ready for shipment. The reject bars are removed from the inspection conveyor by the crane, loaded on to cathode cars and returned to the refining furnace.

The mold press, which is electrically driven, is on the same level as the casting wheel. Ample floor space is left next to the mold press for dressing and storing molds.

Instruments are provided for recording temperatures, pressures and drafts.

A typical furnace charge is composed as shown in Table 9. The charge is melted, skimmed, rabbled with air, poled and cast in the usual manner. The average time for the different operations is given

TABLE 9.—*Furnace Charge and Operating Schedule*

	Furnace Charge, Tons	Operating Time	Hr.	Min.
Cathodes	165.1	Charging	1	28
Starting-sheet scrap	2.1	Melting	12	39
Reject bars	8.2	Skimming		32
Skulls and cleanings	3.3	Blowing	3	58
Discard molds	3.9	Poling	2	0
	———	Casting	3	45
Total	182.6			

in Table 9. During the casting period the temperature of the copper is maintained at about 2025° F. For this control an alumel-chromel thermocouple, enclosed in a chrome-iron protection tube, is placed in the pouring ladle. The temperature of the molds is held at about 215° F. at the pouring point. This temperature is determined by base-metal pyods inserted in holes in the bridges of several molds.

The mold wash consists of three parts by volume of bone ash and two parts by volume of calcined bone black. One part of this mixture

is agitated with five parts of distilled water and applied to the molds with Murphy spray guns.

All of the electrolytic copper produced here has been cast into wire bars. Each bar is rigidly inspected for defects. The percentage of good bars has averaged approximately 93. The average chemical analysis of wire bars for a three-year period shows: copper, 99.9610 per cent; sulfur, 0.0017; arsenic, 0.0005; antimony, 0.0002; iron, 0.0013; nickel, 0.0001; oxygen, 0.0351; silver, 0.019 oz. per ton; gold, trace. Conductivity has been: hard-drawn, 98.44; annealed, 101.25.

The slag from the refining furnace has averaged 0.033 tons per ton of good bars and contained 37.78 per cent copper. It is shipped to the converter plant for treatment.

Operating supplies for furnace and casting operations have averaged as follows per ton good bars: fuel oil, 24.50 gal.; coke, 5.43 lb.; charcoal, 9.15 lb.; poles, 106.67 lb.; pipe, 0.58 ft.; bone ash, 0.416 lb.; bone black, 0.235 lb.

Acid Plant

With the oxide division of the reduction plant operating at full capacity, the sulfuric acid required for leaching purposes amounts to somewhat over 300 tons of 60° Bé. acid per day. The plant for production of this acid is capable of generating at least 375 tons of 60° Bé. acid per day with all three of the sections in operation. It is situated just above and to the east of the converter building, at practically the same level as the sulfide roaster plant and the charge floors of the reverberatory furnaces. It consists of seven calcining units, Cottrell treaters, three acid-generating units and a mixed acid plant.

The available sulfur-bearing material is flotation concentrate. This is calcined in Wedge roasters. The furnace gases pass through flues to Cottrell treaters where the dust is precipitated. The gases from the treaters are conveyed through flues to the Glover towers. Each of the three acid-generation units can produce 125 tons of 60° Bé. acid per day. In these units, packed towers have been substituted for the customary open lead chambers.

A comprehensive description of the Andes sulfuric acid plant was published in the February, 1932, issue of *Chemical & Metallurgical Engineering*. In the following description, operating data and details are omitted.

Roaster Division

The acid roaster plant consists of seven calcining units contained in a steel building. Each unit includes a track bin for receiving concentrate, a charge hopper above the roaster, a seven-hearth Wedge roaster 22 ft. in diameter, a calcine hopper under the furnace, and accessories.

Five of the roasters are equipped with auxiliary Cottrell treaters similar to those in the sulfide roaster plant. They are used only when the furnace gases are being discharged to the air.

The dry concentrate (−1 per cent moisture) is transferred from the drying furnaces in the sulfide roaster plant to the track bins at the acid roasters in covered calcine cars. Each track bin has a 6-in. screw feeder that delivers the concentrate to the boot of a bucket elevator, which in turn discharges into the hopper above the roaster. These hoppers have belt feeders, which advance the concentrate to star feeders discharging upon the top roaster hearth. The calcine from the seventh, or bottom, hearth drops into the calcine hopper and is transferred to the reverberatory plant in cars.

The gases from each furnace pass through a downtake into a balloon-shaped flue parallel to the roaster building. From the balloon flue the gases enter the Cottrell dust precipitators. Three plate and wire-type treaters are provided but usually only two are operated at the same time. To withstand the high temperature they are equipped with chrome-steel electrodes and electrode holders. The exit gases from the Cottrells are conveyed to the Glover towers through brick-lined steel flues. In order to conserve the heat in the furnace gases, the steel shells of the flues and treaters are not only brick lined but have either Silocel brick behind the regular brick lining or insulating material outside the shells. The dust that accumulates in the balloon flue and treaters is drawn off into cars and shipped to the reverberatory plant.

Acid Generation

The three acid-generation units operate in parallel and are independent of each other. Each unit consists of one Glover tower, five reaction towers, three Gay-Lussac towers, fans, coolers, pumps and auxiliary equipment.

The Glover towers function in the customary manner; that is, they denitrate the nitrous vitriol and mixed acid and concentrate the weak acid advanced from the reaction towers. The Glover towers are of the usual construction. The brick walls are sheathed with 10-lb. sheet lead supported on steel frames. The towers are of rectangular section and are 16 by 30 ft. by 50 ft. high inside the lead curtains. The height of the brick packing is 38 ft. The gas delivered to the Glover towers contains approximately 7 per cent sulfur dioxide and has a temperature of 400° to 500° C. It enters each tower through two brick-lined flues and leaves the top through three 30-in. dia. lead flues, which convey it into the top of the first reaction tower of the unit. Each Glover is provided with three acid coolers that operate in parallel.

The five reaction towers in each generation unit take the place of the customary lead chambers. The reactions in the packed towers

are the same as those in the open chambers except that they are much more intensive. These towers are of rectangular section and are 12 by 36 ft. by 53 ft. high. They have no brick walls. The lead walls, or curtains, are supported on steel frames. The height of the brick packing is 41 ft. The flow of gas is alternately down and up. Connections between towers are in each case three 30-in. lead flues. Three coolers, connected in series, cool the acid from the five reaction towers.

The Gay-Lussac towers are operated in the usual manner except that a circulating load of acid is maintained over them in addition to the regular quantity advanced to the Glover tower as nitrous vitriol. These towers are the same size and construction as the reaction towers. The flow of gas is upward in all Gay-Lussac towers. Three 30-in. lead pipes connect adjacent towers. The gas from the last tower in each unit before escaping to atmosphere passes through Cottrell treaters for the removal of acid mist.

Lewis vertical acid pumps are used for transferring acid. For circulation, the acid is pumped to distributing tanks suitably located on top of the units. The acid flows from these tanks to distributors over each tower, which in turn feed splash heads. The acid make, consisting of a mixture of Glover (60° Bé.) and weak (50° Bé.) acids, is pumped to the acid-storage tanks above the leaching vats.

Four hard-lead fans are provided in each unit for moving the gas through the plant.

Mixed-acid Plant

The niter loss suffered in the generation units is made up by introducing mixed nitric-sulfuric acid into the Glover towers. The mixed acid is prepared by decomposing sodium nitrate with sulfuric acid in retorts, which are heated by fuel-oil fires, and condensing the nitric acid vapor in cold 60° Bé. sulfuric acid.

The mixed-acid plant contains four cast-iron nitric acid retorts, each capable of handling 2000 lb. of sodium nitrate per charge, and two absorption towers over which cold 60° Bé. sulfuric acid is circulated. Auxiliary equipment consists of storage tanks, coolers and pumps.

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